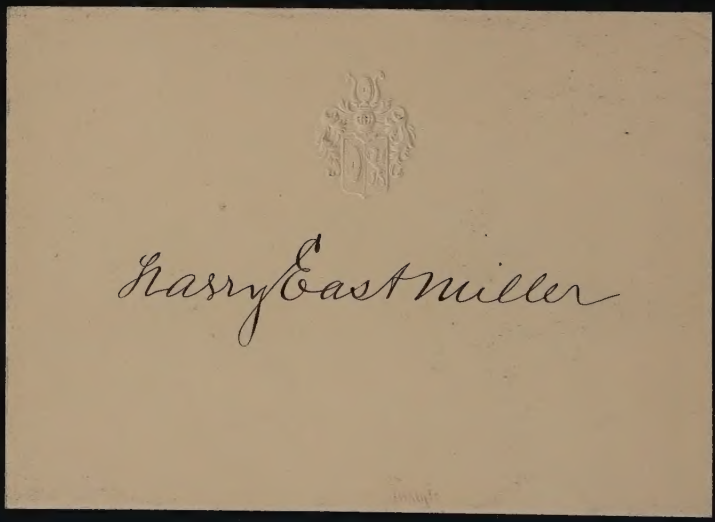
# THE CYANIDE PROCESS OF GOLD EXTRACTION

SECOND EDITION.

JAMES PARK.

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1896.

THE

# CYANIDE PROCESS

OF

# GOLD EXTRACTION.

*A TEXT-BOOK FOR THE USE OF METALLURGISTS AND STUDENTS AT SCHOOLS OF MINES, ETC.*

SECOND EDITION

RE-WRITTEN, ENLARGED AND ILLUSTRATED.

BY

JAMES PARK

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*Authorised Text-book for N.Z. Government Schools of Mines.*

CHAMPTALOUP AND COOPER, AUCKLAND, N.Z.

GEORGE ROBERTSON AND CO., MELBOURNE.

1896.

# PREFACE TO SECOND EDITION.

THE rapid exhaustion of the first edition, which was issued with a Text-book on Practical Assaying and Chemistry in the beginning of last year, has enabled the present edition to be placed before the public in a greatly enlarged form. The whole of the matter has been re-written, and embodies the lectures the author originally prepared for his classes at the School of Mines.

In the hands of different metallurgists and chemists, the cyanide process has made rapid strides during the past few years, more especially in the designing of plants and mechanical appliances to reduce the cost of treatment. At the Witwatersrand Goldfields, it is used on a very large scale for the treatment of tailings, the operations being conducted with much skill and economy under the supervision of noted metallurgical chemists and engineers.

In New Zealand there are some twenty-five cyanide plants, mostly situated in the Hauraki Goldfields. At the greater number of these the ore is dry-crushed and subjected to direct cyanide treatment, but at a few it is wet-crushed, copper-plate amalgamated, and the tailings treated by cyanide, on lines similar to those adopted on the Rand.

The author, in his capacity as Manager of the Government Experimental Works and Cyanide Plant at the Thames for a number of years, had the opportunity of treating many different classes of gold and silver ores, both simple and complex, and he trusts that his experience and practical hints relating to the construction of the plant and the treatment of the ore will be of service to those engaged in using the process. The chapter on the testing, assaying, making up, and dilution, etc., of cyanide solutions, has been specially written in a very simple and concise form for the information of students, and intelligent workmen in batteries and cyanide plants, who wish to qualify themselves in the technical operations, a knowledge of which is essential for the successful working of the process. The manager of a cyanide plant need not be a trained analytical chemist, but he should know sufficient chemistry to be able to modify the treatment to suit varying conditions and classes of ore; and some experience in the working and management of milling machinery will always prove of great service to him.

The author gratefully acknowledges the assistance he has received from the valuable contributions of different writers which have appeared from time to time in the leading mining journals in Europe, America, South Africa, and Australia, and from the works of different authors, to which reference will be found in the text.

THE AUTHOR.

THAMES SCHOOL OF MINES,
May, 1896.

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THE

# CYANIDE PROCESS

FOR THE

# EXTRACTION OF GOLD AND SILVER.

## CHAPTER I.

### McARTHUR-FORREST PROCESS.

It has long been known that gold and silver are soluble in solutions of alkaline cyanides; but it is only within the past few years that this principle has been applied on a commercial scale for the extraction of the precious metals from their ores.

The discovery of the fact that a dilute solution of potassium cyanide is a rapid solvent for natural gold, must rank among the most remarkable discoveries of the present century in metallurgical science; and the widespread and successful application of the fact must mark an epoch in the history of gold extraction.

### SCOPE OF THE PROCESS.

The cyanide process can be applied with success for the treatment of free-milling ores in which the gold occurs in fine particles, or of tailings and concentrates resulting

from wet-crushing and copper-plate amalgamation, or dry-crushing and pan-amalgamation. It can also be used for the treatment of many so-called refractory ores, especially those in which the gold occurs in such a finely-divided form that even amalgamation in pans fails to recover a satisfactory percentage of the values; or of ores in which the gold is coated with a film of metallic oxide or sulphide, rendering it non-amalgamable; or of ores in which the gold is associated with, or entangled in, a highly pyritic matrix.

All the common ores of silver are soluble in dilute solutions of cyanide. Those most readily soluble are the chloride (AgCl) and the sub-sulphide ($Ag_2S$), and these are fortunately the most abundant; but the rate of dissolution of silver and its ores is much slower than that of gold.

## LIMITATIONS OF THE PROCESS.

The cyanide process cannot be applied with success for the treatment of ores in which the gold occurs even in a fairly coarse condition. When an ore contains a proportion of both fine and coarse gold the cyanide process may be used to extract the fine gold, but a supplementary treatment will have to be used to recover the coarse gold, since the slowness of the dissolution would take too long for a commercial basis of working.

With free-milling ores of this class the recovery of the coarse gold is generally effected by copper-plate amalgamation; and, in the case of wet-crushing, this treatment precedes the cyanide leaching, and in the case of dry-crushing it follows it.

The experience gained during the use of the cyanide process has shown that solutions of potassium cyanide,

even when very dilute, act most energetically on all the sulphide, oxide, and carbonate ores of copper, and also on the sulphide of antimony; hence when any of these is present, even in a small proportion, the treatment of the ore becomes difficult, if not impossible, on account of the great consumption of cyanide. In practice it is found that an unduly large consumption of cyanide is generally accompanied by a low rate of extraction of the gold and silver contents.

## CHAPTER II.

### THE CHEMISTRY OF THE PROCESS.

When gold is acted on by an aqueous solution of potassium cyanide, a solution is obtained which, when evaporated, yields octahedral crystals having the composition of the auro-potassic cyanide ($Au\,KCy_2$), which is a double cyanide of gold and potassium.

The exact reaction which takes place when gold is dissolved by potassium cyanide is not yet well understood, being still a subject of much doubt and uncertainty. According to some authorities, the gold is first oxidized before it is dissolved; while others maintain that the cyanide is first oxidized before it acts on the gold.

The reaction suggested by Elsner in 1842 is the one now most generally accepted by chemists. It is represented by the following equation:—

$$2\,Au + 4\,KCy + O + H_2O = 2\,Au\,KCy_2 + 2KHO.$$

According to the above equation, oxygen is necessary for the dissolution of the gold, and this has received substantial support from my own experiments in 1891, and those of Skey* and McLaurin† in 1892 and 1893, in New Zealand.

The valuable researches of Skey and McLaurin have shown that the rate of dissolution of pure gold, under

* Skey, N.Z. Mines Report, 94.
† Trans. Chem. Soc., 1893.

theoretical conditions, reaches a maximum in passing from dilute to concentrated solutions of potassium cyanide. By actual experiment, it was proved that the maximum rate was reached with a 0·25 per cent. solution of cyanide. On a working scale the maximum varies with the character of the mineral constituents of the ore, and can easily be determined by a series of laboratory experiments.

A weak solution is always more active than a strong one, and McLaurin considers this remarkable fact may be accounted for by supposing that the rate of dissolution of gold is partly dependent on the number of cyanide molecules in unit volume; and partly on the number of oxygen molecules in the same volume; and that the solubility of oxygen in cyanide solutions decreases with concentration.

Weak aqueous solutions of cyanide possess a very marked so-called selective action for gold and silver when these metals are associated with ores of copper and antimony. This circumstance becomes very prominent during the treatment of cupriferous ores on a large scale.

**Consumption of Cyanide.** — According to Elsner's equation, about 4·5lbs. of cyanide should dissolve 100 ounces of gold, but in practice it is found that it takes nearly forty times that quantity. The causes which operate in the practice of the process to effect so large a consumption of cyanide, other than required by Elsner's simple equation, are at present not fully investigated.

Potassium cyanide is, chemically, a most active and subtle organic compound, possessing the property of forming so large a number of complicated and unexpected combinations in the presence of mineral acids and base metals, that its reactions and behaviour with different

classes of ore, and under varying conditions, will only be unravelled by much patient research, both in the laboratory and the working plant.

During the treatment of ores by the cyanide process the most puzzling difficulties are continually met with, requiring the constant care and attention of the metallurgist in charge.

**Causes of Loss of Cyanide.**—Some of the principal and more apparent causes of the enormous loss of cyanide which takes place in the practical working of the process are as follows:—

1. Loss by absorption in the vats or tanks.
2. Loss by decomposition by atmospheric carbon dioxide.
3. Mechanical loss in residues, and by dilution of solutions during washing.
4. Loss by decomposition due to mineral acids and salts.
5. Loss due to presence of ores soluble in cyanide.
6. Loss when gold exists as amalgam.
7. Loss due to presence of charcoal.

**Loss by Absorption in Vats.**—This is especially noticeable in new plants. At the Witwatersrand Goldfields, in South Africa, the loss from this cause is said by Mr. C. Butters to amount to a pound of cyanide per ton of tailings treated. At the first monthly "clean up" in a new plant the actual extraction is often twenty or more per cent. under the theoretical, but after a few months it generally reaches within three or six per cent. of the extraction, as determined by assay.

**Loss due to Decomposition by Atmospheric Carbon Dioxide.**—The carbonic acid gas of the atmosphere decomposes potassium cyanide with the formation of potassium carbonate, and the liberation of hydrocyanic (prussic) acid thus :—

$$2KCy + CO_2 + H_2O = K_2CO_3 + 2HCy.$$

The prussic acid thus liberated would be neutralized by any caustic alkali present in the cyanide solution.

**Mechanical Loss in Residues, and Dilution during Washing.**—During washing there is an inability to wash out the whole of the cyanide from the residual tailings. The dilution of the cyanide solutions also occasions a loss of cyanide in washing. A large quantity of dilute cyanide solution is formed, a portion only of which can be utilised to make up fresh solutions.

**Loss by Decomposition due to Mineral Acids and Salts.**—The metallic minerals most commonly found associated with gold in quartz veins are iron pyrites, copper pyrites, zinc blende, galena, and antimonite. By far the most common and abundant of these is iron pyrites.

It has been shown by Skey and others that clean fresh iron pyrites is not acted on by solutions of cyanide. The products of the decomposition of this mineral, however, act most destructively on cyanide, and the obvious conclusion to be drawn from this is that the treatment of pyritic tailings, or concentrates, by cyanide should be undertaken with as little delay as possible.

In the shallow parts of mines, the pyrites is generally oxidized into the ferric oxide, which does not act chemically on cyanide, but causes a mechanical loss through the formation, both in wet and dry crushing, of extremely fine

slimes, which are very absorbent and retentive of cyanide solutions.

Iron pyrites ($FeS_2$) is decomposed by atmospheric oxygen in the presence of moisture into the soluble ferrous sulphate and free sulphuric acid, according to the following equation :—

$$FeS_2 + H_2O + 7O = FeSO_4 + H_2SO_4$$

In the kiln-drying of ores to be dry-crushed, the heat to which the ore, often in large pieces, is subjected, is not very uniform, especially in large kilns. With pyritic ores the sulphides are decomposed at certain temperatures into oxides and soluble sulphates; and at higher temperatures the salts are converted into oxides.

The steam generated from the moisture in the fuel and in the ore itself assists this reaction. In the kiln, where the temperature is high, reducing gases are evolved, and these may impede the oxidation of the sulphides, causing the formation of lower sulphides and basic sulphates, which are insoluble in water but react on cyanide.

As we have seen, the atmospheric oxidation of pyrites results in the production of free sulphuric acid and ferrous sulphate. This ferrous sulphate may in turn be decomposed by the action of the air into insoluble basic sulphates. Thus, partially oxidized pyritic ores or tailings may contain free sulphuric acid, soluble ferrous sulphate, insoluble basic sulphates, and probably also traces of other basic salts of complex and variable composition, all of which react upon solutions of potassium cyanide, thereby causing a loss of cyanogen.

The reactions which are most likely to take place in acid ores or tailings are :—

(*a.*) The liberation of hydrocyanic acid.
(*b.*) The formation of ferro and ferricyanides.

The free acids in the ore react on the cyanide as shown by the equation :—

$$2KCy + H_2SO_4 = 2HCy + K_2SO_4$$

Feldtmann considers it possible for the hydro-cyanic acid thus liberated to diffuse itself through the ore and dissolve appreciable quantities of gold.* For this reason he strongly condemns the practice of washing acid tailings in the leaching vats, as these must always contain a residual portion of cyanide from which prussic acid would be liberated. Any gold dissolved by this gas would be carried away in the water or alkaline wash ; and to avoid this possible source of loss, which he thinks may account for the mysterious discrepancy sometimes found between the assay and the actual extraction, he recommends the system of washing in one vat and leaching in another.

Of the iron salts, the one of most common occurrence in pyritic ores or tailings is the soluble ferrous sulphate ($FeSO_4$), which reacts with potassium cyanide to form potassium ferro-cyanide and sulphate thus :—

$$FeSO_4 + 6KCy = K_4Fe\,Cy_6 + K_2SO_4.$$

The potassium ferro-cyanide thus formed is, in its turn, reacted on by any excess of ferrous sulphate still present with the production of Prussian blue according to the equation :—

$$3K_4Fe\,Cy_6 + 6Fe\,SO_4 + 3O = Fe_2O_3 + 6K_2SO_4 + Fe_4\,(Fe\,Cy_6)_3.$$

* Feldtmann, Notes on Gold Extraction, p. 5.

A blue colour in the solution, or on the surface of the tailings, or in the seams of the staves of the vats, indicates a large consumption and loss of cyanide due to imperfect washing and neutralization of the acidity in the preliminary treatment.

A white scum or precipitate is sometimes seen on the surface of the solutions when they are coming off acid. This precipitate turns into Prussian blue by exposure to air and light.

The normal ferric sulphate $Fe_2(SO_4)3$ is insoluble in water, and cannot be removed by ordinary water-washing. It reacts with potassium cyanide, causing a loss of cyanogen due to the liberation of prussic acid and the formation of the ferric hydrate as shown by the two following equations:—

$$Fe_2(SO_4)_3 + 6KCy = Fe_2Cy_6 + 3K_2SO_4$$

and

$$Fe_2Cy_6 + 6H_2O = Fe_2(HO)_6 + 6HCy.$$

It is probable that in most partially oxidized pyritic ores and tailings the ferrous and ferric sulphates exist together, the former in large excess. In this case the decomposition of the cyanide would result in the production of ferrous cyanide and potassium sulphate thus:—

$$12KCy + 3Fe\,SO_4 + Fe_2\,(SO_4)_3 = Fe_3\,(Fe\,Cy_6)_2 + 6K_2SO_4.$$

In the case of earthy pyritic ores, the weathering or oxidation of the metallic sulphides would result in the production of sulphates of magnesia, lime, or alumina. The action of these sulphates is not very clear, but they most likely react on cyanide with the liberation of prussic acid.

The above reactions clearly emphasize the necessity of a most careful preliminary treatment of pyritic material, in order to avoid undue loss of cyanide and ensure satisfactory results.

All the iron salts and earthy sulphates can be rendered innocuous by the application of an alkali before treating with the cyanide. By this treatment all the soluble iron salts are precipitated as the ferrous hydrate, which rapidly oxidizes to the ferric hydrate; while the basic ones soon oxidize in the presence of the alkali. It is important to remember that the alkali should be applied before, and not with the cyanide solutions, as these iron salts will destroy the cyanide as much in a strongly alkaline as in a nearly neutral solution. When the tailings contain free acid only, alkali and cyanide should be applied together.

**Loss due to Presence of Ores Soluble in Cyanide.**—The sulphide, oxide and carbonate ores of copper, and the sulphide of antimony are acted on by potassium cyanide both in weak and strong solutions, and thereby cause a loss of cyanide in proportion to their abundance in the ore. In the treatment of an ore containing as little as 0·25 per cent. of copper the consumption of cyanide will be doubled.

It is during the treatment of cupriferous ores that the so-called selective action of cyanide becomes most apparent. An ore may contain sufficient copper to decompose a 1 per cent. solution of cyanide and give a low extraction of gold, whereas a 0·5 per cent. solution would dissolve proportionately less copper, and give a fairly satisfactory extraction of the gold. But the same results would be obtained even in the absence of copper, for it has already been shown that the rate of dissolution of gold reaches a maximum in passing from dilute to strong solutions. Hence

a 0·5 per cent. solution should extract more gold than a 1 per cent. solution, the weaker solution being nearer the strength of maximum rate of dissolution which has been proved experimentally to be a 0·25 per cent. on pure gold.

The cyanide treatment of ores and zinc-precipitation of the gold have shown the existence of copper in ores in which no trace of that metal could be detected even by the most rigid chemical examination on large samples. An instance of this came under the notice of the author at the Crown mines at Karangahake. The ore being treated there consisted of almost pure white quartz, free from all metallic impurities; nevertheless a portion of the zinc in the precipitation boxes was often coated with a film of bright metallic copper. The copper could not be derived from an outside source, or from any of the mechanical fittings in the mill or cyanide plant, and Mr. James Napier, the metallurgist and chemist in charge, was of the opinion that it existed in the ore in an infinitesimally small quantity, and only became manifest on the zinc turnings after the treatment of hundreds of tons of ore.

Copper pyrites is oxidized to the soluble sulphate at low temperatures, and this salt requires a greater heat to decompose it than iron pyrites. It is, therefore, probable that a portion at least of this mineral present in an ore, being dried in kilns, preparatory to dry-crushing and direct cyanide treatment, would be sulphatized, and thereby cause an appreciable loss of cyanide in a manner similar to that caused by the decomposition products of iron pyrites.

Malachite and azurite, the green and blue carbonates of copper, are both readily soluble in dilute solutions of cyanide, with the production of copper-potassic cyanide and liberation of prussic acid.

Antimonite, the grey sesqui-sulphide of antimony, is also readily acted on by weak cyanide solutions. It is frequently met with in the gold-bearing ores of the Thames and Reefton goldfields. The presence of a small percentage of antimonite in the large accumulation of tailings at Boatman's Creek, near Reefton, is said to have caused all attempts to treat them to end in failure, chiefly owing to the large consumption of cyanide and the low rate of extraction.

**Loss of Cyanide when Gold Exists as Amalgam.**—It is well known to most millmen that a considerable portion of the gold in tailings, resulting from copper-plate amalgamation or pan-amalgamation, exists in the form of amalgam. When such tailings have to be treated the cyanide has to dissolve the mercury as well as the gold, thus causing a larger consumption of the solvent than would be necessary if the gold existed in a free state.

According to Gmelin, mercury is not dissolved or acted on by potassium cyanide; but the practical working of the cyanide process in New Zealand has shown that his conclusion is contrary to actual experience.

At the cyanide works of the Cassel Gold Extracting Company at Waihi, where a large stack of tailings and residues from pan-amalgamation were treated, 75lbs. of mercury were collected in the condenser attached to the furnace for roasting the zinc slimes. The mercury thus recovered was only a small proportion of the mercury dissolved by the cyanide, and afterwards precipitated in the zinc-extractor with the bullion. On every occasion when the roasting of the zinc slimes was being conducted, so much mercury was volatilized that the vapours pervaded every part of the buildings, condensing on every cool

surface, and amalgamating all objects of gold and silver worn by the workmen.

The volatilization of mercury during the roasting of zinc slimes, resulting from the treatment of tailings, has been noted by the author on several occasions at the Thames School of Mines Experimental Cyanide Works, and is of frequent occurrence at the cyanide plants at Kuaotunu. The same circumstance was noted by Dr. Scheidel at the Sylvia Cyanide Works at the Thames,* where mercury was found in the zinc-bullion in considerable quantities.

The mercury generally occurs in the tailings in the form of amalgam in a very fine state of subdivision, and is dissolved by the cyanide, together with the associated gold and silver. It is precipitated with the bullion in the zinc precipitation boxes. When the zinc slimes are oxidized the greater portion of the mercury is volatilized.

**Loss of Cyanide Due to the Presence of Charcoal in the Ore.**—It has long been known to chemists that charcoal possesses the property of decomposing auro-potassic cyanide solutions, and many attempts have been made to substitute charcoal for zinc precipitation. The author has found that a very large bulk of charcoal and prolonged contact with the solutions are required to effect complete precipitation of the bullion. Charcoal also, after some hours' contact, acts on potassium cyanide solutions, thus causing a loss of cyanogen. For these reasons it seems improbable that charcoal will ever supersede zinc as a precipitant of gold from cyanide solutions.

Although charcoal has not yet been successfully substituted for zinc, still its presence in ores that have been

* The Cyanide Process, Dr. A. Scheidel, p. 35.

dried and dry-crushed preparatory to cyanide treatment may cause a loss of both cyanogen and gold during the leaching.

At the Kapai-Vermont Cyanide Works, at Kuaotunu, New Zealand, the metallurgist, Mr. J. A. Walker, states that it pays to employ labour to pick the lumps of charcoal and partially-carbonised wood out of the ore before it passes to the ball-mill to be pulverised. This is a subject well worthy of further investigation.

**The Action of Cyanide on Metallic Sulphides.**—This was the subject of an investigation by Mr. William Skey, analyst for the New Zealand Government. The results were communicated to the Government early in 1895, and published in an interesting paper in the Annual Report of the Mines Department for 1895.

Referring to the oft-repeated statement that very dilute solutions of potassium cyanide have no effect on copper sulphides, it will be seen that Mr. Skey's results entirely disprove this. They are as follows:—

1. *Chalcopyrites*, crushed, then thoroughly well washed, was afterwards kept in a cyanide solution of 0·03 per cent. for one hour. The filtered solution contained a very perceptible quantity of copper as cyanide, also traces of sulphur and oxidized compounds of sulphur.

2. *Copper-glance*, crushed and washed, then subjected to the cyanide solution of 0·03 per cent. for one hour. The filtered solution gave the same reaction as in the case of chalcopyrites.

3. *Covelline* (sub-sulphide of copper). As previously shown, this sulphide is very easily attacked by weak solutions of cyanide, sulpho-cyanide of copper resulting.*

* Vol. xxi., Trans. N.Z. Inst., 1888, "On the Preparation of Artificial Chromes."

The 0.03 per cent. solution dissolves the mineral as a sulpho-cyanide and cyanide of copper.

The carbonates and silicates of copper, as they naturally occur even—that is, in the dense form—are also decomposed by cyanide of this strength.

These results take in all the compounds of copper that are of general occurrence at the gold-mines, and they show that potassic cyanide, however weak, will decompose copper ores, generally, when in contact with them.

Mr. Skey's results upon certain other ores that are frequently associated with gold in reefs are as follows:—

*Stibnite* (sulphide of antimony).—This ore is generally supposed to be unaffected by potassic cyanide, but was found to be very easily and largely affected by this salt. The sulphur of this mineral, like the sulphur in certain copper sulphides, shows a great tendency to combine with potassium to form that most objectionable salt—sulphide of potassium, the rest of the sulphur combines to form sulpho-cyanogen.

*Galena* (sulphide of lead). — This mineral is slowly attacked by cyanides, but all its sulphur combines with cyanogen to form the harmless compound, sulpho-cyanogen, and lead comes into solution combined with that radical.

*Zinc-blende* (sulphide of zinc).— Hardly affected by cyanide of any strength, it is commonly held to be perfectly neutral thereto; but a strip of silver, buried in the crushed ore, was blackened in one hour, showing that a slight decomposition had taken place.

*Iron-pyrites* (bi-sulphide of iron).—Almost, if not quite, unaffected by cyanide solutions of any strength.

*Sulphur* (free).—The Government Geologist states that he met with sulphur in a free state in tailings at Boatman's Creek, in the Reefton district, as a product of the decomposition of stibnite.

It is asserted by Wiggers* to be insoluble in potassic cyanide solutions, but it is very easily soluble therein if only the air, with which it is generally charged, be driven off, say, by boiling water.† As in the cases of the solution of copper, this means a loss of solvent power for gold and silver.

Cold solutions were used throughout Skey's experiments.

The formation of alkaline sulphides during the cyanide process, as applied to ores containing stibnite and copper-sulphides, is, undoubtedly, the chief cause of the loss of gold that so often occurs in that process.

With oxidized ores, as malachite or dioptase, a loss of available cyanide will necessarily occur; but with these sulphides, in addition to loss of cyanide, there will be a loss of gold, and a still greater loss of silver, in proportion to the quantities present. This loss is brought about by the sulphur—that is, the alkaline sulphide—sulphurising these metals to form sulphides with them,‡ and the sulphide film so formed upon the metal prevents, or greatly retards, the proper action of the cyanide solution.

That gold does combine, and very readily, with the sulphur of both the alkaline and hydrogen sulphide Skey has already shown. §

* Ann. Ch. Pharm., Vol xxxix., page 319
† Vol. iv, Trans. N.Z. Inst., page 330.
‡ Vol. iii., Trans. N.Z. Inst., page 216
§ Vol. xxi., Trans. N.Z. Inst., 1888.

It is to precipitate the sulphur that gets into the cyanide in the cyanide process that Mr. McArthur has proposed to use, or does use (as per patent), a soluble lead salt dissolved in the cyanide.

The problem for the chemist at the cyanide works is to find a practical method whereby all the sulphur of antimonial and cupreous sulphides can be made to combine with the cyanogen rather than with the potassium of the cyanide.

The following results obtained by Skey show how extremely objectionable alkaline sulphides are, when present in the cyanide solution:—

A rather strong solution of the cyanide, containing a small proportion of sulphur, was placed over a strip of gold coupled with a piece of copper-glance (sulphide of copper), but no solution of gold was perceived; however, on substituting the copper-glance with chalcopyrites, the gold was rapidly removed.

This experiment shows that the gold was sulphurised at the outset by the alkaline sulphide present in the cyanide, and that it required connecting with a substance of a very negative kind in order to effect the decomposition of the auriferous sulphide so formed.

Further experiments of a different kind showed that while pure cyanide of 1 per cent. solution dissolved a given weight of gold in ten minutes, a solution, of the salt of the same strength, but containing $\frac{1}{100000}$ part of sulphur (as a sulphide),* required two hours to dissolve the same weight of gold. The speeds were as 1 to 12 in favour of the pure cyanide.

* Vol. xxi., Trans. N.Z. Inst., 1888, "On the Preparation of Artificial Chromes."

The following results show how very much even a gentle sulphuretting, or flouring of the gold, interferes with its solution:—

Gold sulphurised 60 seconds in $K_2S$, dissolved in cyanide in 62 minutes.

Gold sulphurised 54 seconds in $K_2S$, dissolved in cyanide in 50 minutes.

Gold sulphurised 1 second in $K_2S$, dissolved in cyanide in 36 minutes.

Gold, clean, dissolved in cyanide in 12 minutes.

The gold was well washed from adherent potassic sulphide before being placed in the cyanide. Making clean gold the unit, the approximate times of dissolution are 1, 3, 4, and 5.

**The Action of Manganese Oxides on Cyanide.**—During the treatment of a parcel of ore from the Komata gold-mine, near Waitekauri, the author found there was an unusual consumption of cyanide. The ore consisted of soft mullocky, friable quartz, coloured quite black with a large percentage of pyrolusite and wad.

A series of experiments were afterwards made to determine the cause of the loss, and the results of these led the author to the conclusion that the manganese oxides oxidized a portion of the cyanide into cyanate. It is well known that pyrolusite parts with a portion of its oxygen under the influence of heat alone, but more readily so in the presence of an easily oxidizable substance.

The author's results are so far only of a tentative character, but they point to the oxidation of cyanide into cyanate, as a probable cause of loss of available cyanide.

## CHAPTER III.

### LABORATORY EXPERIMENTS.

The cyanide process is essentially a chemical one, and a commodious and well-equipped laboratory forms one of the most important and necessary parts of the whole plant.

It is the duty of the metallurgist in charge to determine by actual experiment the lowest strength of cyanide solution to extract an adequate percentage of the gold, and also to devise means of overcoming the problems which are inseparable from the treatment of different classes and grades of ore, with so active and subtle a compound as potassium cyanide.

The testing and valuing of the ores before, during, and after treatment must be entrusted to a careful and trustworthy assayer. The testing and making up of the working solutions are very simple operations, that may be left to experienced and intelligent workmen who possess a knowledge of arithmetic extending as far as decimals.

With free-milling gold ores the actual working extraction will generally be as high as that obtained in the laboratory, but too much reliance must not be placed on the laboratory experiments in the case of ores containing copper or antimony.

The author's experience is that high extractions may be obtained in the laboratory from ores totally unsuited for treatment by the cyanide process on a working scale.

The conditions on the one hand are theoretical, on the other actual, and before adopting the cyanide treatment for a sulphide or mineralized ore, working experiments should be made on parcels ranging from two to five tons, in order to ascertain the consumption of cyanide and actual extraction. If the working trials are successful the cyanide treatment may be adopted with confidence.

On the other hand, in the case of an ore containing comparatively coarse gold, the laboratory experiments—where the sample is *hand-crushed*—will give lower results than those obtained in practice in the cyanide works. The author made a number of experiments on an ore from Marlborough. The average extraction in the laboratory was under 40%, while the cyanide plant extracted over 60%. At the battery the ore was dry-crushed through a 60-mesh screen, and investigation showed that a large portion of the gold was reduced fine enough to pass through the screen, and thus became amenable to cyanide treatment.

## THE ACTUAL EXPERIMENT.

1. Procure six bell-jars, about four and a-half inches in diameter. When bell-jars are not procurable, clear pint glass bottles, with the bottoms cut off, will answer the purpose quite well. In the neck of each jar fit a cork, perforated with one hole. Through the hole pass a short length of glass tube, on the end of which place a few inches of pliable black rubber tubing. On the end of the rubber tubing place a screw-clip, by means of which the rate of percolation of the cyanide solutions can be regulated to a nicety.

2. Now, invert the jars, and fix them in a wooden frame so as to stand upright. In each jar place a thin layer of small rounded pebbles, about the size of French

beans; above the pebbles place an inch of coarse sand, and above this, half-an-inch of fine sand. Above the fine sand place a piece of loose scrim, the size of the jar. This completes the filter bed.

When a large number of cyanide experiments are being made, a box divided into three compartments, to hold the three grades of material for the filter bed, should be kept well replenished and near at hand.

3. Next procure a fair sample of the pulverised ore to be tested, weighing, say, six or eight pounds. Mix thoroughly, and carefully assay to accurately determine the original value.

Check assays should always be made, and if there is a serious discrepancy between the assay and its check, amounting to over 3 per cent. of the value, fresh assays should be made. The assays form the basis of the calculations and final results of the experiments, and hence the greatest accuracy should be aimed at.

When the ore to be tested is from the battery, or mill, it should be placed in the jar in the condition it comes from the dust-bin, except, of course, when the tests are to determine the degree of fineness which would give the best economic extraction.

When the ore is hand-pulverised, a separate portion should be reduced to pass through, say, a 30-mesh, 40-mesh, and 60-mesh sieve respectively. Separate tests should be made of each grade so as to determine to what extent the extraction is affected by the varying fineness of the ore.

4. Introduce into each jar 10 or 12 ounces of the powdered and sampled ore, the value of which has been

obtained by careful assay. Mark the jars numbers 1, 2, 3, 4, 5, and 6.

5. In the case of tailings or ores containing iron pyrites, or other base metallic sulphides, the samples in the jars should be washed once or twice with clean water to remove any soluble sulphates. With very acid tailings a very dilute alkaline wash may be applied.

6. No general rule can be laid down as to the strength of the cyanide solutions to be used, as this will depend as much on the character of the sample as on its value. All the ores of copper and antimony, and the oxides of manganese, act on and use up cyanide, and when any of these is present a stronger series of solutions will have to be tried than in the case of clean ores. With the latter a useful series of solutions would contain :—

0·25 %, 0·3 %, 0·35 %, 0·4 %, 0·45 %, 0·5 % of cyanide.

With pyritic ores or tailings, or those containing copper, antimony, or manganese compounds, the most instructive series would be :—

0·4 %, 0·45 %, 0 5 %, 0·55 %, 0·6 %, 0 65 %.

It is necessary with every new ore to make a number of laboratory experiments to ascertain the proper strength of cyanide solution to extract an adequate proportion of the gold and silver contents.

7. To each jar, already charged with the ore, add the same weight of solution as of ore. The excess of solution is required because a large portion immediately finds its way into the filter bed. Record the numbers of the jars and the strength of cyanide used in each. Regulate the screw-clips so that the percolation will take at least thirty hours.

A longer time may be tried if the first trials are not successful. If the solution comes through too quickly return it again and allow it, this time, to percolate more slowly.

8. When the leaching is complete, wash with two washings of clean water, allowing the wash-water to drain as rapidly as possible. The washing is complete when the wash-water no longer gives an alkaline reaction.

9. Test the strength of the spent solution to ascertain the percentage of cyanide used. The solution and washings are collected and measured together, then tested for cyanide.

The consumption of cyanide can be calculated from the ratio of the bulk of solution and washings to that of the original solution.

On a working scale the consumption of cyanide is generally much less than that shown by the laboratory experiments.

Sometimes the cyanide and different washings are kept separate and evaporated down with the addition of litharge (in the manner described under The Assay of Cyanide Solutions) and the gold actually extracted by each, calculated separately. The results afford an instructive lesson on the value of successive washings.

10. Remove the leached and washed pulp from the jars, and dry, mix thoroughly, and again assay. Then calculate the percentage of extraction from each jar by difference, recording the results and assay values as follows:—

| | Original Value. ozs. | dwts. | grs. | After Leaching. ozs. | dwts. | grs. | Percentage of Recovery. |
|---|---|---|---|---|---|---|---|
| Gold .. | 2 | 4 | 12 | 0 | 4 | 12 | 89·9 |
| Silver .. | 1 | 0 | 4 | 0 | 3 | 6 | 83·9 |
| Value .. | £9 | 0 | 0 | £0 | 18 | 4 | 89.8 |

The calculation is simply a matter of proportion. As an example take the recovery of gold.

| | ozs. | dwts. | grs. | |
|---|---|---|---|---|
| Original gold .. | 2 | 4 | 12 | per ton |
| After leaching .. | 0 | 4 | 12 | ,, |
| Extracted .. | 2 | 0 | 0 | ,, |

Then if two ounces were extracted from 2ozs. 4dwts. 12grs., what would the extraction be from 100 ?

2ozs. 4dwts. 12grs. : 2ozs. :: 100 : 89·9 per cent.

$$\frac{2 \times 100}{2{\cdot}225} = 89{\cdot}9 \text{ per cent.}$$

An easy and expeditious method of calculating the percentages of extraction in the laboratory-test is to use the weights of the bullion, gold, and silver (in grains or grams) as the basis of computation instead of the same extended as ounces, dwts., and grains.

*Example :—*

| | Original Assay. | After Leaching. | Extracted. | Recovery°/₀ |
|---|---|---|---|---|
| Bullion .. | ·0020 | ·0002 | ·0018 | 90·0 |
| Gold .. | ·0018 | ·0001 | ·0017 | 94·4 |
| Silver .. | ·0002 | ·0001 | ·0001 | 50·0 |
| Value | £9 2 0 | £0 10 3 | £8 11 9 | 94·3 |

*The calculation : —*

For Bullion.

$$\frac{\cdot0018 \times 100}{\cdot002} = 90$$

For Gold.

$$\frac{\cdot0017 \times 100}{\cdot0018} = 94{\cdot}4$$

For Silver.

$$\frac{\cdot0001 \times 100}{\cdot0002} = 50$$

For Value.

$$\frac{8.587 \times 100}{9\cdot1} = 94\cdot3$$

11. Compare the results obtained and adopt the strength which gives the highest extraction.

Remarks.—With a series of experiments it will be found that the percentage of extraction, or rate of dissolution of the gold, reaches a maximum with a cyanide solution of a certain strength, and that above and below this strength the rate of extraction rapidly diminishes. The strength of cyanide solution which dissolves the maximum percentage of gold will depend on the character of the ore. The so-called selective action of cyanide is not so apparent in the laboratory experiments as it is in practice.

On a working scale it soon becomes evident in the treatment of base sulphide ores that a strong solution of cyanide of potassium dissolves a large proportion of the base metals and a small proportion of the gold, while a weak solution dissolves a large proportion of the gold and a small proportion of the base metals.

## CHAPTER IV.

### TO TEST, MAKE UP, AND ASSAY CYANIDE SOLUTIONS.

**To Test the Strength of Cyanide Solutions.**—This is an operation of great simplicity, and can be performed with accuracy and expedition by any intelligent workman by a volumetric method of estimation. The standard solutions should always be made up under the personal supervision of the chemist in charge of the works.

Three different volumetric methods may be used for the determination, namely:—

1. By standard solution of silver nitrate.
2. By standard solution of mercuric chloride.
3. By standard solution of iodine.

**By Silver Nitrate Standard Solution.**—This is the method generally adopted in cyanide plants. It is a modification of Liebig's volumetric estimation of cyanogen.

The reaction depends on the fact that when a solution of silver nitrate is added to a solution of potassium cyanide, the cyanogen unites with the silver, appearing as a white precipitate, which is immediately dissolved by any free KCy, which may still be present, forming a double cyanide of potassium and silver.

This is shown by the equations:—

$$AgNO_3 + KCy = AgCy + KNO_3 ;$$

and

$$AgCy + KCy = AgKCy_2.$$

A standard solution of silver nitrate can thus be made up from the molecular weights as follows :—

$AgNO_3$ saturates 2KCy.
170 = 130
17 = 13

With grams use a decinormal solution; then if 17 grams of silver nitrate are dissolved in 1000c.c. of water, 1c.c. will be equal to ·013 grams of KCy.

To Make Silver Nitrate Standard Solution.—Take 17 grams of silver nitrate, triple crystallised if procurable, and dissolve in one litre (1000c.c.) of distilled water. In large works, where much testing is going on, it is advisable to dissolve 34 grams in two litres; then place in stoppered-bottle and mark.

To Test Solutions of KCy :—

1. Fill a burette with silver nitrate solution.

2. Measure 13c.c. of cyanide solution to be tested from another burette and transfer to a small beaker.

3. Run in cautiously $AgNO_3$ standard solution from the burette till the white precipitate which appears just *ceases* to *re-dissolve*, when the beaker is shaken; that is, when a faint permanent opalescence appears the reaction is complete.

4. Read off number of c.c. of standard solution used and divide by 10. The result will represent the percentage of available KCy. For example :—

Suppose 13c.c. of KCy sol. took 14·5c.c. of $AgNO_3$, then :—

$$\frac{13{\cdot}5}{10} = 1{\cdot}35\ \%\ \text{of KCy.}$$

If a strong solution is being tested, in order to save $AgNO_3$, measure off, say, 3c.c. or 4c.c. of cyanide solution instead of 13 and titrate with standard. Thus, if 4c.c. required 6c.c. of silver nitrate, 13c.c. would require 19·5; and 19·5 divided by 10 = 1·95 % KCy.

Even greater accuracy may be obtained in testing strong solutions, such as those in the dissolving tank, together with a saving of silver nitrate, by measuring off 13c.c. of the strong solution and diluting with water to 130c.c. Then measure off 13c.c. of this diluted solution and titrate with silver nitrate as described above. Note the number of c.c. of standard solution required to complete the reaction and this will represent the percentage of KCy in the strong solution, for since the 13c.c. of diluted solution contained only a tenth of the original 13c.c. of strong solution, there is hence no need to divide the quantity of silver nitrate by ten.

To test the strength of very dilute cyanide solutions measure off 130c.c. of the solution, titrate with silver nitrate, and divide the number of c.c. of standard required by 100, and the result will give the percentage of available KCy, thus:—

130c.c. of cyanide solution required 5c.c. of standard, then:—

$$5 \div 100 = 0{\cdot}05\ \%\ \text{KCy}.$$

To avoid calculation and reduce the liability to make mistakes in reading the burette, a standard solution of silver nitrate can be made up by dissolving 13·07 grams* of silver nitrate in 1000c.c. of water. To test a cyanide solution with this measure off 10c.c. and titrate with silver nitrate. Note the number of c.c. of standard required to complete

* Thus:—13 : 10 :: 17 : 13·07

the reaction, divide by ten, and the result will be the percentage of available KCy, thus if :—

10c.c. of cyanide solution required, 5c.c. of silver nitrate

$$5 \div 10 = 0{\cdot}5\ \%\ \text{KCy}.$$

Remarks.—The addition of a few drops of a solution of potassium iodide to the solution to be tested renders the end reaction more defined and reduces the danger of over-estimating the KCy due to the akalinity of the solution.

Two burettes should always be used, one to measure the cyanide solution, and one for the silver nitrate standard. The gram burette should be graduated to 1-10th gram. Erdmann floats should always be used so as to obtain the exact measure.

## To Test Cyanide Solutions with Grain Standard Solution.

1. When grain burettes are used make up a normal solution of silver nitrate by dissolving 170 grains in 10,000 grains of pure water.

2. Measure off, from a burette, 130 grains of cyanide solution to be tested.

3. Fill a burette with the silver nitrate standard.

4. Run silver nitrate into cyanide solution until the white p.p. which at first forms just ceases to re-dissolve. Note the number of grains required.

5. The number of grains of silver nitrate solution used to titrate divided by 100 will give the percentage of available KCy.

6. For example, 56 grains of silver nitrate were used, then :—

$$\frac{56}{100} = 0{\cdot}56 \text{ per cent. of KCy in the solution.}$$

ESTIMATION OF KCy BY MERCURIC CHLORIDE (HANNAY).

(1.) When a solution of mercuric chloride is added to a solution of potassium cyanide, a cyanide of mercury is formed, but is at once dissolved by any excess of KCy present.

When all the free or available KCy has been used a bluish-white opalescence of $HgCy_2$ appears if a slight excess of mercuric chloride is added. This permanent opalescence indicates the end of the reaction.

(2.) To MAKE UP STANDARD SOLUTION.— Use the equation :—

$$HgCl_2 + 2KCy = HgCy_2 + 2KCl.$$

271 saturates 130.

27·1 = 13 in a decinormal solution.

From the above molecular weights, dissolve 27·1 grams of mercuric chloride in 1000c.c. of distilled water ; then 1c.c. will equal ·013 grams of KCy. Place in a stoppered-bottle and mark.

(3.) THE ACTUAL DETERMINATION—

(*a.*) Fill a burette with mercuric standard.

(*b.*) From another burette measure off 13c.c. of the cyanide solution to be tested, and to this add about 3c.c. of semi-dilute ammonia.

(*c.*) Now run in mercuric standard very cautiously, with constant shaking, until a permanent bluish-white opalescence is produced.

(*d.*) Note the number of c.c. of standard required to complete the reaction; divide this number by 10, and the result will be the percentage of available KCy present, thus, if :—

6·5c.c. were required to complete titration, then :—

$$6{\cdot}5 \div 10 = 0{\cdot}65\ \%\ KCy.$$

Remarks.—With pure substances, this reaction is very delicate, but with cyanide solutions, containing much impurity, it is not so reliable as the silver nitrate method. Caustic alkalis do not interfere with the reaction. The author has made a number of simultaneous tests, with working cyanide solutions, by the silver nitrate and mercuric chloride methods, and the results obtained were practically the same throughout.

ESTIMATION OF KCy BY IODINE STANDARD.

(1.) This method depends on the fact that when a solution of iodine is added to one of potassium cyanide, the iodine loses its colour so long as any undecomposed cyanide remains.

(2.) To Make up Iodine Standard Solution.—Use the reaction :—

$$2I + KCy = KI + ICy.$$

254 saturates 65.

25·4 = 6·5 in a decinormal solution.

Therefore, to make a standard, weigh out 25·4 grams of iodine, place in a beaker with 200c.c. of water and add sufficient potassium iodide to completely dissolve the iodine with frequent shaking.

When the iodine is dissolved, make up to 1000c.c. with pure water, and place in a stoppered-bottle. Then :—

1 c.c. = ·0065 grams KCy.

(3.) The Actual Determination—

(*a.*) Fill a burette with the iodine standard.

(*b.*) From another burette measure off 6·5c.c. of cyanide solution to be tested, and to this add carbonic acid (20c.c. of ordinary soda-water

will do) to convert the caustic and mono-carbonate alkalis, contained in all commercial cyanide, into bi-carbonates.

(*c.*) Now run in iodine standard, cautiously and slowly, until a slight but permanent yellow colour is produced.

(*d.*) Read off the number of c.c. of standard required, divide by 10, and the result will be the percentage of KCy required.

Remarks.—This method does not give reliable results in the presence of sulphides, or when the cyanide solution is muddy or discoloured.

Some Useful Constants.

The cubic content of a circular vat in feet =
Dia$^2$ × ·7854 × depth in feet.
1 cubic foot of water = 62½lbs. nearly.
1 ton of water contains about 36 cubic feet.
1 gallon equals 10lbs.
1 lb. avd. equals 7000 grains troy.

**To Make Up Cyanide Solutions.**—There are two different methods in common use in cyanide plants. In some cases the requisite amount of solid cyanide salt is added to the sump solution; and in others, and perhaps more frequently, the working strength is made up by adding strong solution from the dissolving tank to the sump solution.

The following exercises will render these methods clear:—

1lb. of pure KCy dissolved in 100lbs. water gives a 1 per cent. solution; therefore, if you have a vat containing

100 cubic feet of water to make up to say 0·6 per cent. you would require 37·35lbs. of pure KCy.

$$\text{Thus } 100 \times 62\tfrac{1}{4} = 6225\text{lbs. of water,}$$

and if 100lbs of water require 0·6lbs. KCy, 6225lbs. would require :—

$$100 : 6225 :: 0{\cdot}6 : x$$

$$\frac{6225 \times 0{\cdot}6}{100} = 37{\cdot}35\text{lbs.}$$

Commercial cyanide is seldom pure; you would, therefore, have to use a greater quantity to make up the required strength.

Suppose the crude KCy equals 78 per cent. of KCy, then :—

$$78 : 100 : 37{\cdot}35 : x$$

$$\frac{100 \times 37{\cdot}35}{78} = 48\text{lbs. crude KCy.}$$

The same form of calculation will do for making up any required quantity of cyanide solution. Suppose 4ozs. of a 0·5 per cent. solution were required.

Then if 100ozs. of water require 0·5ozs. of cyanide, how much would 4ozs. require ?

$$100 : 4 :: 0{\cdot}5 : x$$

$$\frac{4 \times 0{\cdot}5}{100} = 0{\cdot}02 \times 480 = 9{\cdot}6 \text{ grains.}$$

If you have a 0·2 per cent. solution and you wish to make it up to, say, 0·5 per cent., subtract the 0·2 per cent. already in the solution from 0·5 per cent., leaving 0·3 per cent. required. Then proceed to make up as directed in the preceding paragraph.

(1.) I have 4000lbs. of sump-solution containing 0·2 per cent. of available KCy, which I wish to make up to a

0·5 per cent. solution, how much additional KCy will be required?

Then :— 100 : 4000 :: 0·3 : $x$

$$\frac{4000 \times \cdot 3}{100} = 12\text{lbs. pure KCy.}$$

If your crude KCy salt contains only 82 per cent. of KCy, then :—

$$\frac{12 \times 100}{82} = 14{\cdot}6\text{lbs. crude KCy required.}$$

(2.) How many lbs. of solid cyanide salt of 75 per cent. strength should be used to make up 10 tons of a 0·4 per cent. working solution?

Ans.—119·46lbs.

(3.) How many lbs. of solid cyanide salt of 82 per cent. strength should be used to make up 5 tons of a 0·45 per cent. working solution, using a sump-solution containing 0·15 per cent. of KCy for making up?

Ans.—40·97lbs.

(4.) How many lbs. of a 14 per cent. stock cyanide solution should be used to make up 10 tons of a 0·4 per cent. working solution, using a 0·18 per cent. sump-solution for making up?

*Solution.*—This is easiest determined by an Alligation, the tea-mixers rule of proportion, thus :—

Strong solution 14·00 ·40 working solution
Working solution ·40 ·18 sump-solution

Proportion of weak = 13·60 + ·22 = proportion of strong

Clear of decimals, then :—

1360 + 22 = 1382.

Here we have 1382 parts or lbs. of the required mixture, containing 22 of strong and 1360 of weak solution; therefore, if 22lbs. of the strong solution give 1382lbs. of the required mixture, how many lbs. will be required for 10 tons of the mixture?

$$1382 : (10 \times 2240) :: 22 : x$$

$$\frac{22400 \times 22}{1382} = 356{\cdot}58$$

The answer is, therefore, 356·58lbs.

(5.) How many lbs. of a 22 per cent. stock solution should be used to make up 9 tons of a 0·5 per cent. working solution, using a 0·12 sump-solution for making up?

Ans.—351·27lbs.

(6.) How many lbs. of a 12 per cent. stock solution should be used to make up 10 tons of a 0·6 per cent. working solution, using a 0·15 per cent. sump-solution for making up? Before drawing from the stock solution first utilize 4 tons of a 0·8 solution already in the solution vat.

Ans.—359·74lbs.

*Solution.*—First find out how much of the working solution can be made up from the 4 tons of 0·8 per cent. cyanide solution, thus:—

| | | | |
|---|---|---|---|
| Strong solution | ·80 | ·60 | working solution |
| Working solution | ·60 | ·15 | sump-solution |

Proportion of weak = ·20 + ·45 = proportion of strong

Clear of decimals, then:—

20 + 45 = 65 of required mixture.

Then, if 45 of the strong (0·8 per cent.) give 65 of the required mixture, the 4 tons already in the solution tank will give 5·77 tons, thus :—

$$45 : 4 :: 65 : x$$

$$\frac{4 \times 65}{45} = 5{\cdot}77.$$

And 10 — 5·77 = 4·23 tons to be made up from the 12 per cent. stock solution, thus :—

Strong solution 12·00 — ·60 working solution
Working solution ·60 — ·15 sump-solution

Proportion of weak = 11·40 + ·45 = proportion of strong

Clear of decimals :—

1140 + 45 = 1185 of mixture.

Now, if 45lbs. of the stock solution (12 per cent.) give 1185lbs. of the required mixture, how much will give 4·23 tons?

$$1185 : (4{\cdot}23 \times 2240) :: 45 : x$$

$$\frac{4{\cdot}23 \times 2240 \times 45}{1185} = 359{\cdot}74\text{lbs.}$$

## TO DILUTE CYANIDE SOLUTIONS.

(7.) How many tons of a 0·45 per cent. working solution would 6 tons of a 0·8 per cent. solution of cyanide make, using water for dilution?

Ans.—10·66 tons.

*Solution.*—

$$·45 : ·80 :: 6 : x$$

$$\frac{80 \times 6}{45} = 10{\cdot}66.$$

(8.) How many tons of a 0·4 per cent. working solution of cyanide would 8 tons of a 0·6 per cent. solution make, using a 0·12 per cent. sump-solution for dilution ?

Ans.—13·71 tons.

*Solution.*—

| | | | |
|---|---|---|---|
| Strong solution | ·60 | | ·40 working solution |
| Working solution | ·40 | | ·12 sump-solution |
| Proportion of weak = | ·20 | + | ·28 = proportion of strong |
| | 20 | + | 28 = 48 of mixture. |

Now, if 28 of the strong solution give 48 of the required mixture, 8 tons will give 13·71 tons, thus :—

$$28 : 8 :: 48 . x$$

$$\frac{8 \times 48}{28} = 13{\cdot}71 \text{ tons.}$$

(9.) How many tons of a 0·6 per cent. cyanide solution would 8 tons of a 0·7 per cent. solution make, using a 0·2 per cent. sump-solution for dilution ?

Ans.—10 tons.

## To Test the Strength of Crude KCy.—

Commercial KCy is formed when any nitrogenous organic bodies, such as hoofs, clippings of hides, wool, and blood are fused with potassium carbonate. This product is very impure, and is lixiviated in a vessel containing finely divided metallic iron, yielding the yellow prussiate of potassium ($K_4F_6Cy_6$), which is the starting point of all cyanogen compounds.

Crude cyanide of potassium is formed by the action of heat on the yellow prussiate, thus :—

$$K_4FeC_6N_6 = 4\,KCN + FeC_2 + N_2.$$

The chief impurities in commercial cyanide are black carbide of iron, alkaline carbonates, and sometimes alkaline chlorides and sulphides in small quantities.

To accurately test the strength of the solid cyanide salt, for the free or available KCy which it contains, proceed as follows:—

(1.) Break a cake of KCy in two, and select a piece, say, a pound in weight showing the whole thickness of the cake in section.

(2.) Reduce this pound to a coarse powder, sample well, and further pulverize to a moderately fine powder.

(3.) Weigh out 1 gram of powdered and sampled KCy.

(4.) Dissolve in pure water and make up to 100c.c.

(5.) Measure off 13c.c. of this solution and titrate with silver nitrate standard solution from a burette as previously described. Note number of c.c. of standard required to form a permanent p.p.; divide by 10 and this will give the amount of KCy in 1 gram of the crude salt.

For example: Suppose 13c.c. of KCy solution required 7·5c.c. of standard, then:—

$$\frac{7{\cdot}5}{10} = {\cdot}75 \text{ KCy in 1 gram,}$$

which is equal to 75 per cent. of KCy in the crude salt.

### The Assay of Cyanide Solutions.—

(1.) Measure half-a-pint of solution and evaporate to a small bulk in a round iron drying-dish, over a

Bunsen-flame, or on the furnace-lid. As the evaporation proceeds rub the sides down so as to collect the whole of the dissolved salts at the bottom.

(2.) To the solution add 600 grains of litharge. Mix well; evaporate cautiously to dryness.

(3.) Then transfer to clay-crucible and mix with 200 grains of glass-powder, 100 grains of soda, and 48 grains of argol. Cover with a little borax, and fuse. When fused, pour and allow to cool.

(4.) Cupel the lead-button and weigh the resulting bead of bullion; then part so as to determine the weight of gold and of silver; then refer to the table at the end of the chapter to ascertain the quantity of each per ton of solution.

When the resulting gold is weighed with gram weights refer to the Gram Table; and, when in grains, to the Grain Table.

Remarks.—When a large number of determinations have to be made, ordinary tin-plates form efficient evaporating dishes; in this case also the litharge can be stirred into the solution before the evaporation begins.

## CROSSE'S METHOD OF TESTING CYANIDE SOLUTIONS FOR GOLD.

(1.) Measure half-a-pint of cyanide solution, and add silver nitrate solution until a precipitate ceases to form. The silver salt should be added a little at the time, and the solution well shaken after each addition. All the gold in the solution is precipitated as the argentic auric cyanide.

(2.) Allow the precipitate to settle; decant off the clear solution; dry the precipitate and mix with 200 grains

litharge, 100 grains glass-powder, 100 grains soda, and 48 grains of argol. Fuse, pour, and cupel the lead-button.

(3.) Extract the bead of bullion from the cupel, flatten, and part without weighing.

(4.) Weigh the resulting gold and calculate the results.

This method is more expeditious than the evaporation with litharge, and gives perfectly accurate results for gold. When the determination of the silver in a cyanide solution is required the first method must be used.

**The "Shaking Test" for Consumption of Cyanide.**—This method is much used in the laboratories of the Cassel Cyanide Company and affords a rapid and fairly approximate estimate of the consumption of cyanide with different classes of ore. It is useful for comparative purposes, and as a preliminary means of determining the most suitable strengths of cyanide solutions for laboratory experiments.

(1.) Take 200 grams of the ore and place in a stoppered bottle with 100c.c. of a 0·5 per cent. cyanide solution, for example, and shake for 20 minutes.

(2.) Allow contents of bottle to settle; draw off a portion of the clear solution with a pipette and test for KCy. If it contains a 0·2 per cent. of KCy then 0·3 per cent. has been consumed or decomposed.

(3.) When much cyanide is used up, test the ore for acidity by Feldtmann's method given below.

**To Test Ores and Tailings for Acidity.** —

(1.) Weigh out 224 grams of ore and shake up with 250c.c. of water in a tall glass-jar or cylinder.

(2.) Fill a burette with a standard solution of soda and titrate the ore-solution in the jar until the reaction is neutral to test (litmus) paper.

(3.) Every c.c. of the soda solution used will represent 0·1lb. of caustic soda to be added to every ton of ore (or tailings) in a wash before the cyanide treatment.

To Make Standard Soda Solution. — Dissolve 10 grams (or 154·3 grains) of caustic soda in 1000c.c. of pure water and place in a secure bottle.

1c.c. = 0·1lb. caustic soda.

Remarks.—During the titration, the litmus paper should be dipped in clean pure water from time to time to remove the adhering particles of ore so that the reaction may be clearly seen.

**Tests for Alkaline Sulphides in Cyanide.**— Alkaline sulphides act injuriously in cyanide solutions during leaching, and it is important to detect their presence. They are all soluble in water.

*First Test:*—To the clear cyanide solution add a little acid. If an alkaline sulphide be present sulphur will be liberated, imparting a cloudy appearance to the solution.

*Second Test:*—In the clear solution place a clean bright silver coin. It will become black and tarnished if a sulphide be present.

*Third Test:*—The most delicate test is by means of the nitro-prussides. These are formed by adding a little nitric acid to a solution of ferro or ferri-cyanide of potassium.

Add a little solution of a nitro-prusside to the cyanide solution. If an alkaline sulphide is present the solution will assume a deep brilliant purple colour.

FOR THE ASSAY OF CYANIDE SOLUTIONS.

| If ½-pint of Solution give of Fine Metal. | One Ton of Solution will give Fine Metal. | | | If ½-pint of Solution give of Fine Metal. | One Ton of Solution will give Fine Metal. | | |
|---|---|---|---|---|---|---|---|
| Gram. | ozs. | dwts. | grs. | Gram. | ozs. | dwts. | grs. |
| ·0001 | 0 | 0 | 5·5 | ·0200 | 2 | 5 | 20 |
| ·0002 | 0 | 0 | 11 | ·0300 | 3 | 8 | 18 |
| ·0003 | 0 | 0 | 16·5 | ·0400 | 4 | 11 | 16 |
| ·0004 | 0 | 0 | 22 | ·0500 | 5 | 14 | 14 |
| ·0005 | 0 | 1 | 3·5 | ·0600 | 6 | 17 | 12 |
| ·0006 | 0 | 1 | 9 | ·0700 | 8 | 0 | 10 |
| ·0007 | 0 | 1 | 14·5 | ·0800 | 9 | 3 | 8 |
| ·0008 | 0 | 1 | 20 | ·0900 | 10 | 6 | 6 |
| ·0009 | 0 | 2 | 1·5 | ·1000 | 11 | 9 | 4 |
| ·0010 | 0 | 2 | 7 | ·2000 | 22 | 18 | 8 |
| ·0020 | 0 | 4 | 14 | ·3000 | 34 | 7 | 12 |
| ·0030 | 0 | 6 | 21 | ·4000 | 45 | 16 | 16 |
| ·0040 | 0 | 9 | 4 | ·5000 | 57 | 5 | 20 |
| ·0050 | 0 | 11 | 11 | ·6000 | 68 | 15 | 0 |
| ·0060 | 0 | 13 | 18 | ·7000 | 80 | 4 | 4 |
| ·0070 | 0 | 16 | 1 | ·8000 | 91 | 13 | 8 |
| ·0080 | 0 | 18 | 8 | ·9000 | 103 | 2 | 12 |
| ·0090 | 1 | 0 | 15 | 1·0000 | 114 | 11 | 16 |
| ·0100 | 1 | 2 | 22 | 2·0000 | 229 | 3 | 8 |

## GRAIN TABLE.

FOR THE ASSAY OF CYANIDE SOLUTIONS.

| If ½-pint of Solution give of Fine Metal. | One Ton of Solution will give Fine Metal. | | | If ½-pint of Solution give of Fine Metal. | One Ton of Solution will give Fine Metal. | | |
|---|---|---|---|---|---|---|---|
| Grains. | ozs. | dwts. | grs. | Grains. | ozs. | dwts. | grs. |
| ·001 | 0 | 0 | 3·5 | ·060 | 0 | 8 | 23 |
| ·002 | 0 | 0 | 7 | ·070 | 0 | 10 | 11 |
| ·003 | 0 | 0 | 11 | ·080 | 0 | 11 | 23 |
| ·004 | 0 | 0 | 14·5 | ·090 | 0 | 13 | 10 |
| ·005 | 0 | 0 | 18 | ·100 | 0 | 14 | 22 |
| ·006 | 0 | 0 | 21·5 | ·200 | 1 | 9 | 20 |
| ·007 | 0 | 1 | 1 | ·300 | 2 | 4 | 18 |
| ·008 | 0 | 1 | 4·5 | ·400 | 2 | 19 | 16 |
| ·009 | 0 | 1 | 8 | ·500 | 3 | 14 | 14 |
| ·010 | 0 | 1 | 12 | ·600 | 4 | 9 | 12 |
| ·020 | 0 | 3 | 0 | ·700 | 5 | 4 | 10 |
| ·030 | 0 | 4 | 12 | ·800 | 5 | 19 | 8 |
| ·040 | 0 | 6 | 0 | ·900 | 6 | 14 | 6 |
| ·050 | 0 | 7 | 11 | 1·000 | 7 | 9 | 4 |

## CHAPTER V.

### THE APPLIANCES FOR CYANIDE EXTRACTION.

The appliances in use are much the same at all cyanide plants, but their size, shape, and arrangement are subject to endless variations, being chiefly affected by local conditions, the character of the material to be treated, and the individual taste or fancy of the metallurgist. In all cases the designer should utilize the natural advantages at his disposal; and, where possible, the solution vat, leaching vats, vacuum-cylinder, and storage tanks, zinc extractors, and sumps should be placed on four separate tiers or platforms, so as to permit the circulation of the solutions by gravitation.

The necessary appliances for a successful and well-equipped cyanide plant, where the ore is to be treated by percolation, are as follows:—

1. A dissolving tank
2. Solution vats
3. Leaching or percolating vats
4. Vacuum-cylinder and air-pumps
5. Storage vats
6. Zinc precipitation boxes
7. Sumps
8. Solution pump
9. Assay office and laboratory.

**Dissolving Tank.**—This is constructed of wood, iron, or steel. When made of wood, the staves are of 2in. or 2½in. pine; and when of iron or steel, the plates are from ⅛in. to ¼in. thick, according to the size of the vat, and stiffened with angle-iron and hoops. The size varies from 3ft. to 6ft. in diameter, and from 2½ft. to 4ft. in depth.

In large cyanide works a perforated tray, to hold the solid cyanide salt, is suspended over the tank by a chain or steel-wire rope, running over a pulley fixed to a beam overhead. The end of the rope or chain passes over a second pulley on the same beam, fixed at a sufficient distance to permit a balance-weight on the end of the rope to clear the side of the tank.

In practice the solid salt is taken out of the original packing case and cleaned, by removing all adhering particles of sawdust or other packing material with a husk-broom. It is then broken into small pieces and placed in the perforated tray, which is allowed to subside into the solution. The rapid dissolution of the cyanide salt can be effected by imparting motion to the tray by pulling at the weighted end of the rope.

The discharge-hole from the dissolving tank should always be placed three or four inches above the bottom, so as to allow a settling space for the impurities contained in the cyanide.

The impurities contained in commercial cyanide consist principally of black carbide of iron, and other insoluble matters, which would, if permitted, obstruct the solution pipes and choke up the filter webs, and thus cause vexatious delays. Besides this, carbide of iron decomposes the potassic cyanide solution of gold; hence its presence in the solutions would tend to cause a loss by precipitating a portion of the gold.

**Solution Vats.**—These are used for making up the cyanide solutions to the working strength. They are open circular tanks from 14ft. to 20ft. in diameter, and from 4ft. to 14ft. in depth. They are generally constructed of well-seasoned pine. The sides and bottom are made of 6in. planks, 2½in. or 3in. thick. The edges of the side planks are cut radial to the arc of the circle of the vat.

The bottom planks are bolted and dowelled together independent of the sides. The bottom is rebated into the side planks, which are kept tight by round iron hoops, ¾in. to 1½in. in diameter, having three or more cast-iron turn-buckles on each hoop. The side planks, or staves, are kept in their places by the pressure of the hoops alone, which are generally placed from 15in. to 18in. apart, with an extra hoop at the bottom. The hoops are placed, on very large deep tanks, only six or seven inches apart at the bottom, where the pressure is greatest, the distance apart gradually increasing to 18in. or 20in. at the top. The extra hoop is placed as close to the bottom hoop as the turn-buckles will permit.

One or more solution vats may be required according to the size of the plant.

**Leaching or Percolating Vats.**—These are made of many different shapes, sizes, and kinds of material. At first, small square tanks of wood were used, but the difficulty of keeping these tight led to the adoption of circular vats, which are stronger.

In South Africa, Australia, and New Zealand the favourite material is wood. The construction of the circular wooden leaching vats is in every respect the same as that of the solution vats already described, differing only in being provided with discharge-doors. The large vats in use at

the cyanide works of the Waihi Gold and Silver Mining Company are 22½ft. in diameter and 4ft. deep. They are built of 6in. kauri planks, 3in. thick, and bound together with five hoops of round iron, three of which are ⅞in. in diameter, and two 1in., having three turn-buckles on each hoop. Five inches is taken off the depth by the false bottom, filter-frame, and cloth.

At the Simmer and Jack Cyanide Works, Johannesburg, the leaching tanks are constructed of pine. They are 42ft. in diameter and 14ft. deep ; and bound together with fifteen hoops of round iron. They rest on piers of solid masonry. The staves and bottoms are made of 9in. by 3in. material.

At the "Main Reef" works there are six leaching vats, each 26ft. inside diameter, with 8ft. staves, and holding 135 tons of tailings.

The staves are 4½in. wide and 3in. thick, and planed to the bevel by machines, and afterwards hand-dressed on the abutting edges. They are checked ¾in. to fit on the bottom, with a chime 6in. below the check.

The bottom of the vat is made of 9in. by 3in. deals, planed by machine and grooved ⅜in. by ½in. by a drunken saw, and is also hand-dressed on the edges. Clear-pine tongues, 1in. by ⅜in., fill the grooves. The joists across the tunnel, below the vat, consist of 9in. by 3in. deals, bolted together in pairs, and laid 2ft. 3in. apart from centre to centre.

These joists are first laid in position, then the bottom of the vat is laid down, cramped up, and the circle struck out. The bottom is now sawn to the circle, and when bevelled all round is ready for the staves, which are driven up as tightly as possible.

Six hoops of round iron are used to keep the staves in their places; the top pair 1in. in diameter, the middle pair 1⅛in., and the lowest pair 1⅜in. in diameter, with screwed ends.

Each hoop is made in three sections, rolled to the required curve, and connected by cast-iron turn-buckles.

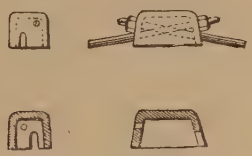


FIG 1. SHOWING CONSTRUCTION OF TURN-BUCKLE.
Scale: ½in. = 1ft.

The screwed ends pass through the turn-buckles, and while each hoop is being drawn up it is hammered with a heavy sledge-hammer.

Two carpenters, practised at the work, can dress the material for a vat, 28ft. in diameter and 8ft. deep, in about a week, and erect it in about four days.

With large vats, constructed of brick and cement, in an excavation in the ground, there is no means of ascertaining what leakage is going on; and, in a process in which gold solutions are being dealt with, an exceedingly small leak, in the course of the year, would represent a considerable loss. For this reason their construction cannot be recommended where material is procurable for the construction of wooden vats.

At the Langlaagte Estate Company's Cyanide Works, S.A., the tanks are round and constructed of brick, faced with hydraulic cement. Their size is 40ft. in diameter and 10ft. deep. At the Crown Reef works the tanks are also of

brick, 40ft. square and 10ft. deep. At the Waihi-Silverton works, N.Z., the tanks are constructed of $\frac{3}{16}$in. steel, being 16ft. in diameter and 4ft. deep. At the Cripple Creek Gold Exploration Company's works, in Colorado, the vats are made of iron, being 20ft. in diameter.

For the direct treatment of dry-crushed ore, the leaching vats are seldom over 4ft. deep, on account of the difficulty of percolation with a greater depth of ore; but, with tailings comparatively free from slimes, the depth varies from 8ft. to 14ft.

The leaching vats, on account of the enormous weight they hold, must be built on strong firm foundations, so as to prevent settling, and the leakage which would be sure to follow. In South Africa they are often built on piers of stone; and in New Zealand and Australia, where timber is plentiful, on massive frames of wood. Whatever the foundations, there should always be free access to the bottom of the tanks, so as to be able to detect and repair leaks.

Each tank is provided with a separate drain-pipe, 1½in. or 2in. in diameter, with two stop-cocks near each other, one over the strong solution launder or pipe, the other over the weak solution launder, leading to their respective zinc-extractors.

The sizes of the pipes for charging the vats with the solutions are as follows:—

| | |
|---|---|
| Vats, 20—24ft. in diameter - - | 2½in. |
| ,, 24—32ft. ,, - - | 3in. |
| ,, 32—40ft. ,, - - | 4in. |

Two wooden launders running parallel with the line of leaching vats afford the simplest, most economical, and effective method of collecting the solutions as they percolate

from the vats. This system enables the solutions from each vat to be tested separately and readily, and by this means any mishaps can at once be detected.

Instead of having stop-cocks on the end of the drain-pipe from each vat, a short length of rubber-hose is sometimes fixed on the end; and by moving the hose the solution can be drained into the strong or weak launder as required.

Steel and iron vats are largely used in America, and are now coming into use in New Zealand. They possess many advantages over wooden ones. They are generally coated with a composition consisting of a mixture of coal-tar, pitch, and kauri gum.

The filter-frame in steel vats is supported on a ring of angle-iron rivetted to the side about 3in. from the bottom. The filter-webbing is laid on the frame and kept in its place by means of a ring of angle-iron, which is constructed in four, six, or eight pieces, or lengths, so as to be easily handled. The pressure of the ring alone is sufficient to keep the filter-cloth tight.

**Filter Frames.**—The old filter-beds of gravel and sand have been entirely superseded by light wooden frames, over which are placed filter cloths or webs, consisting of either extra strong Hessian, loose canvas, cocoa-matting, or burlap. For the filtration of slimes, or dry-crushed ores, which always contain a large percentage of very fine sands, a webbing of strong Hessian or canvas is used; and for tailings or concentrates a webbing of cocoa-matting or burlap.

At the Waihi Cyanide Works the filter frames, designed by the general manager, Mr. H. P. Barry, consist of narrow laths placed parallel, and about an inch apart. On these

laths are nailed, transversely, narrow moulding-like laths, also about an inch apart. An open frame-work or grating is thus obtained, having openings an inch square.

At the Main Reef Cyanide Works, Johannesburg, the filter-frame is made of 3in. by 1in. slats, placed on edge, 6in. apart, their ends being kept 1in. from the sides of the vat. Strips of wood, 1in. square, are nailed on the top of the slats, 1in. apart, to form a support for the cocoa-matting.

The filter frames for large vats are constructed in sections. The sections, when fitted together, form a circular frame about an inch less in diameter than that of the vat. This leaves an annular space between the frame and the vat, which permits the filter-cloth to be firmly grouted in its place by means of a small rope passing round the circumference of the vat.

The author has used such filter frames at the Thames School of Mines cyanide plant for over two years, and finds that they possess many advantages over the old gravel-filters.

**Vacuum-Cylinder and Air-Pump.**—Filtration is generally assisted by creating an artificial vacuum below the filter-bed. The means mostly adopted in New Zealand and Australia to produce a vacuum is an air-tight boiler, or cylinder, connected with an air-pump.

The cylinders are generally constructed of $\frac{1}{4}$in. boiler-plate, with $\frac{3}{8}$in. ends. They are made of different sizes, according to requirements, from 6ft. to 13ft. in length, and from 3ft. 6in. to 6ft. in diameter. They are provided with a solution-guage, vacuum-guage, air-cock, and man-hole.

The air-pump is single or double-acting, with a 6in. or 8in. stroke, making from 80 to 120 strokes per minute, and

capable of producing a vacuum of 26in. of mercury in the cylinder. To prevent heating of the valves it should be surrounded with a water-jacket, through which a current of cold water can continually circulate when the pump is running.

All the stop-cocks, valves, pipes, and connections about the cylinder, air, and solution pumps, and tanks, which are subject to cyanide solutions, should be of black iron.

When the vacuum-cylinder becomes full, the solution is discharged into a storage vat, from which it slowly drains through the zinc-extractor. In order to give timely warning when the cylinder became full, a simple electrical contrivance was used by Mr. Arthur Wilson, the manager, at the Cassel Gold Extracting Company's tailings plant at Waihi.

A small Erdmann float, with a platinum wire fused into the top and coiled into a flat helix, was placed in the solution guage-tube. Two platinum wires were also fused into the upper end of the guage-tube, projecting into the tube, opposite to each other, but not in contact. The platinum wires were connected with a small Leclanche battery, and when the float rose in the solution-guage to the platinum wires, metallic contact was established, and an electric bell in the circuit sounded an alarm.

**Discharge of Leached Residues.**—Where there is a plentiful supply of water, with a good head, the easiest and cheapest method of discharging the residues from the leaching tanks is to sluice them out by a side-door. At the cyanide plant of the Waihi Gold Mining Company the residues are sluiced out by two 2in. hose-pipes under a head of 150ft., giving a pressure of 65lbs. to the square inch.

At the Witwatersrand Goldfields, where there is a scarcity of water, and often a want of fall for the sludge,

the "bottom discharge" is largely practised, the residues being shovelled through a hole in the bottom of the vat into a truck immediately below. At the Barrett Company's works the tailings are shovelled into a launder below the vat, and a stream of water carries them away.

At the Crown, Woodstock, Talisman, Waihi-Silverton, Waitekauri, and Kapai-Vermont Cyanide Works in New Zealand, where the ore contains a percentage of coarse gold, the residues are slowly sluiced over extensive amalgamated copper-plates placed immediately below the discharge holes.

At the Langlaagte Company's Cyanide Works, near Johannesburg, the residues are discharged from the large brick leaching tanks by means of steam-travelling cranes, which lower the bodies of the empty trucks into the tanks, where they are filled by Kaffir labour. When filled the trucks are raised and placed on their carriages, to be wheeled away to the dump.

**Discharge Doors.**—When side-discharge by sluicing is used, one outlet is generally provided for each vat; but in the case of bottom-discharge there are two, four, six, or eight discharge openings to each vat, according to its size.

At the Witwatersrand goldfields the bottom-discharge is employed for discharging the round wooden leaching vats. When filling a deep tank with tailings, a length of wrought-iron pipe or cylinder, three or four feet long, is placed over each discharge-hole, and then the tailings are dumped in. The pipe raises the outlet within a few feet of the surface, and thus facilitates the discharge.

On these fields, Butters' bottom-discharge doors are largely used. Fig. 2 shows their construction.

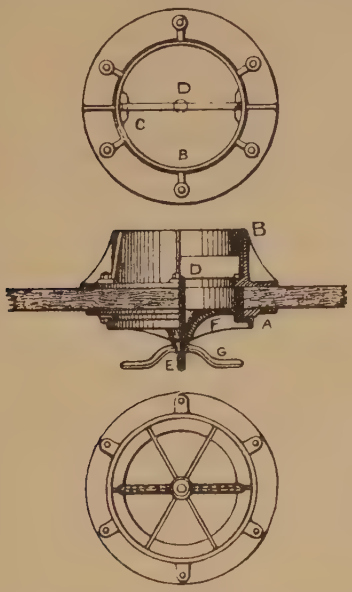



FIG. 2. BUTTERS' BOTTOM-DISCHARGE DOOR.
Scale: $\frac{1}{2}$in. = 1ft.

The arrangement is very simple and effective. On the bottom side of the tank a cast-iron ring, A, is bolted to the cast-iron cylinder, B, inside the tank. Inside the cylinder is a projecting lug, C, upon which rests the hanger, D, which forms part of the screw, E. The cast-iron cover, F, when placed in position, is fastened by the butterfly-nut, G, and by screwing this firmly the whole arrangement becomes watertight. The faces of the ring and cover should be planed perfectly even, so as to make a good joint. The joint is also made tight by a luting of clay.

Fig. 3 shows another bottom-discharge door, designed by Mr. W. F. Irvine. It is simple in construction and not likely to get out of order, but would be more convenient in shallow than deep vats.

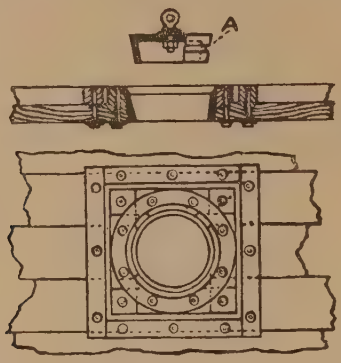



Fig. 3. Irvine's Bottom-Discharge Door.
A—Recess for Packing. Scale: $\frac{1}{8}$in. = 4in.

Fig. 4 shows a modification of a side-discharge door for wooden vats, designed by Mr. W. R. Feldtmann, of Johannesburg.

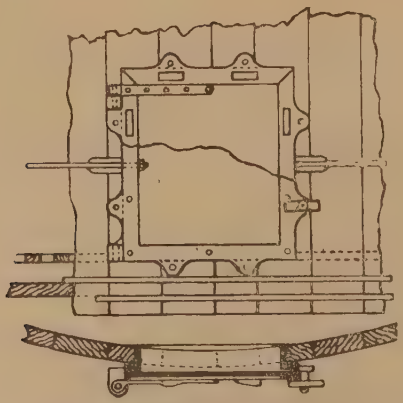
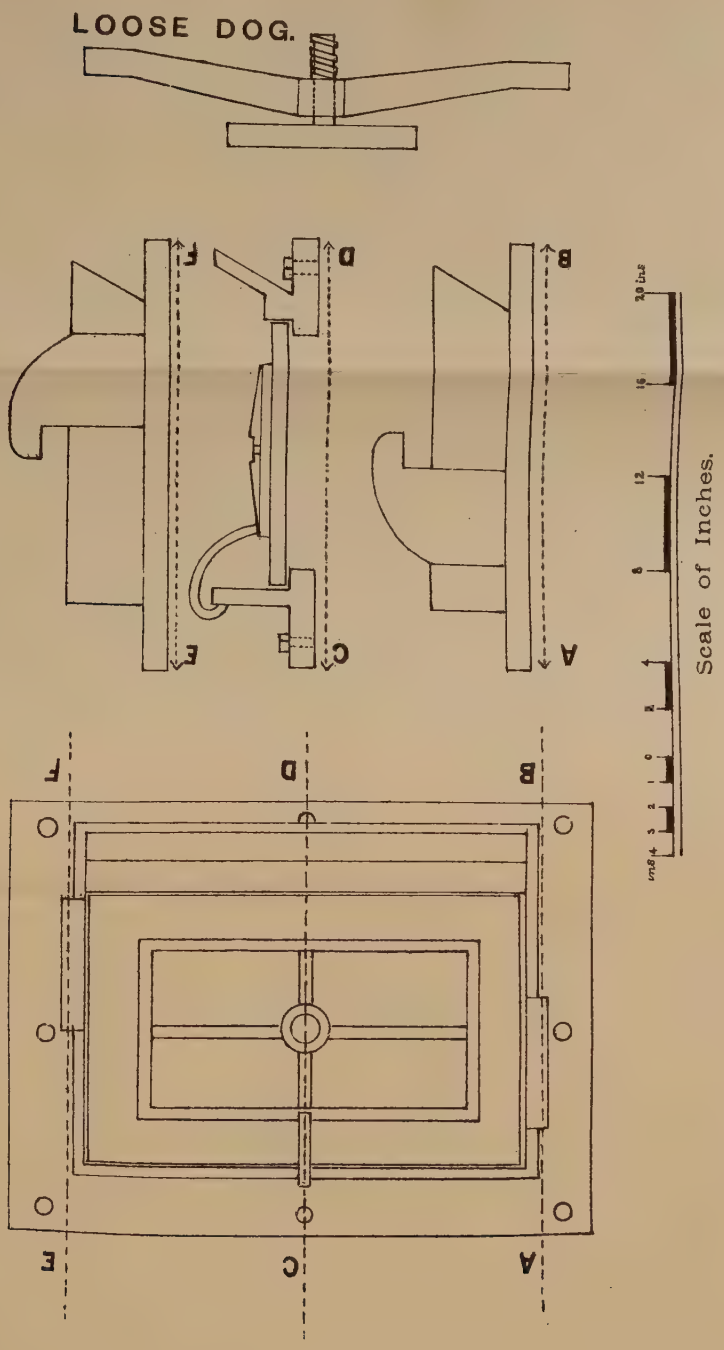


Fig 4. Side-Discharge Door.
Scale: $\frac{3}{4}$in. = 2ft.

# SLUDGE DOOR.



Another method of side-discharge, designed by Mr. Irvine, is in use at the Crown Reef Company's fine cyanide works, where the large square brick and cement vats are provided with doors which permit of the ingress of the discharging trucks.* The door frames are bolted to the cement walls, and the plate-iron doors are drawn tight against these by means of an ingenious arrangement of sliding lugs, bolts, and nuts.

The side-discharge sludge doors used in New Zealand are of the simplest construction and yet perfectly efficient. They consist of a cast-iron frame with two projecting lugs, one at each side, and a projecting bar running along the top side. The lug on the right side is placed with the notch upwards, that on the left with the notch in the reverse position.

The opening is closed by a cast-iron door, which is kept in position by the pressure of a screw acting through a loose iron dog, the ends of which fit into the lugs so as to obtain the necessary leverage.

The door is suspended in front of the opening, preparatory to fixing up, by a hook of bent round iron, which is supported on the projecting bar on the frame. It is rendered watertight by a facing of rubber insertion, fixed on with tar, or by a luting of clay.

These doors seldom give any trouble. They are easily opened or closed by a few turns of the screw. The different parts are shown to scale on Plate I.

At the fine dry crushing and cyanide plant designed and recently erected at the Waihi-Silverton mine by Mr. H. H. Adams, there are twelve leaching vats of $\frac{3}{16}$in. steel,

* Feldtmann, Notes on Gold Extraction, 1894, p. 3.

16ft. in diameter and 4ft. deep, and provided with a central discharge. The bottoms are slightly concave, the depression being 6in. in the radius of 8ft. The discharge hole is 9in. in diameter, and on the outside there is rivetted a length of bent pipe 14in. long, and provided with a door similar to that shown on Plate I. A piece of canvas is placed over the discharge hole, and the weight of the ore pressing on the filter-webbing, which is cut at the centre, prevents it from sagging or slipping. The general arrangement is shown to scale on Plate Ia.

**Sumps.**—There are generally two of these in every cyanide plant, to receive the cyanide solutions after passing through the zinc-extractor, one for the strong solution and one for the weak. In plants dealing with acid ores or tailings, besides these there is often an additional tank for storing the alkaline wash-solutions.

The size of the storage sumps depends on the size of the plant. In most plants in New Zealand they are the same size as the leaching vats. They are constructed either of steel, cement, bricks faced with cement, or wood. The latter is the favourite material. The construction of wooden sumps is the same as that of the storage tanks or vats.

In many plants the sumps are placed below the level of the lower or extractor floor of the cyanide building, and in such cases they are decked over with planks having a man-hole for repairs or cleaning. The depth of the solution is indicated by the tell-tale.

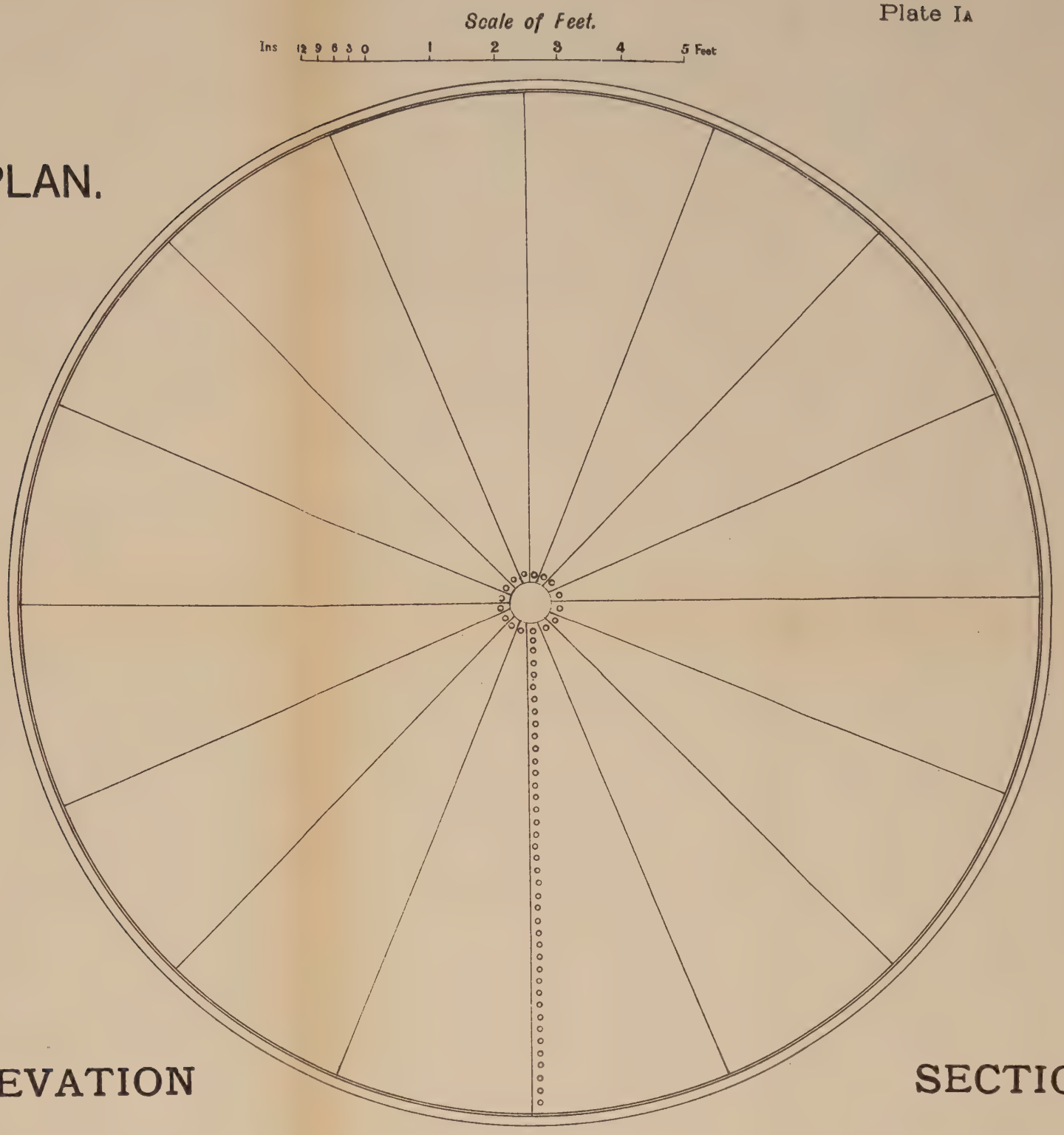
In some tailings plants the working cyanide solution is made up in the strong solution sump. This saves the construction of a storage vat, but is not convenient, and could not be used with advantage for the treatment of dry-crushed ores in which it is necessary to apply the strong solution

# WAIHI-SILVERTON GOLD MINING Co.

## STEEL PERCOLATING VAT.

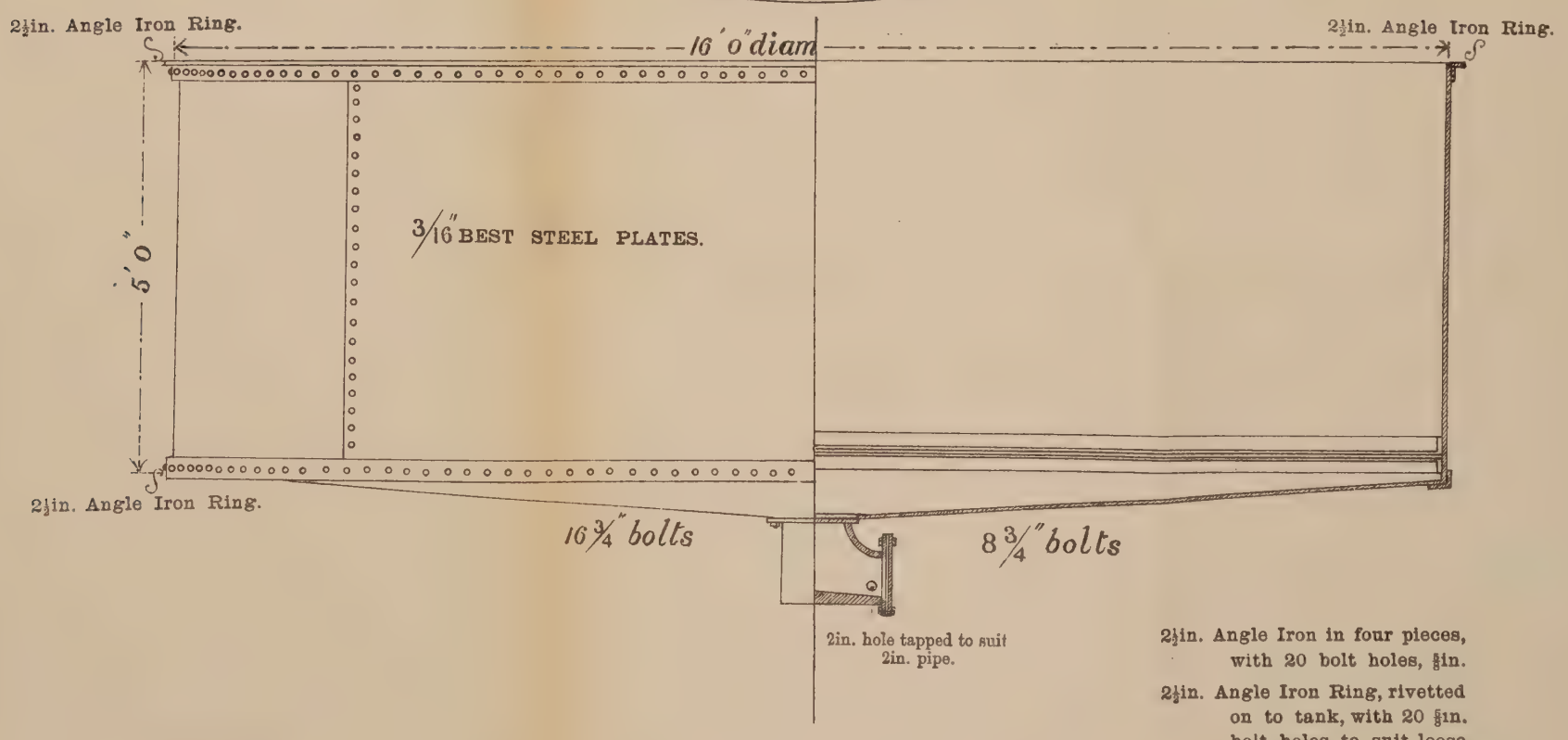
Plate IA

PLAN.




ELEVATION

SECTION.




2½in. Angle Iron in four pieces, with 20 bolt holes, ⅝in.

2½in. Angle Iron Ring, rivetted on to tank, with 20 ⅝in. bolt holes to suit loose flange.

2½in. Angle Iron Ring.

slowly from below, so as to prevent the formation of lumps and channels in the pulp as would be formed were the solution turned on to the dry ore from above.

**Zinc-Extractors.**—There are at least two in every plant, one for the strong solutions and one for the weak. They are constructed of wood, and consist of oblong boxes, each divided into a number of compartments, generally eight, ten, or twelve, by means of partitions and baffle-boards, fixed in such a way that the solution finds its way upwards through the zinc-turnings in each compartment.

The first and last divisions are often filled with sand to act as filters; the first to remove any sandy matter from the solution, the end one to prevent the escape of any fine gold slimes.

The remaining divisions are provided with removable shallow trays of wood, with convenient handles, often of bent round iron, to enable them to be easily lifted up when necessary. The trays support, at the bottom, a wire screen of $\frac{1}{8}$in. mesh, which permits the free circulation of the solutions, and enables the gold and zinc slimes, as they form, to fall through into the bottom of the compartment.

The more recent extractors are cleaned-up from plug-holes on the side, one to each compartment. To facilitate the "clean-up," the bottoms of the compartments slope to the side, and also to the lower side-corner, where the discharge-hole is situated.

A launder of wood or iron is fixed on the discharge-side, immediately under the plug-holes, so as to receive the slimes when they are washed out of the boxes. The top of the extractor, as well as the side launder, is protected with a lid or close-fitting grating of wood or iron, provided with

locks, to prevent the contents of the extractor being tampered with.

To facilitate the withdrawal of the wooden plugs from the discharge-holes, short lengths of rubber-hose are fixed in the holes. The rubber is yielding, and, while rendering the holes watertight, enables the plugs to be withdrawn without the force which is generally necessary when they are driven into the bare holes.

The extractor should be constructed of well-seasoned pine, the sides of 1½in. or 2in. boards, and the divisions of ¾in. boards, well-dressed. The size will depend on the capacity of the plant, and will vary from 12ft. to 20ft. in length, from 2ft. to 3ft. in depth, and from 15in. to 45in. in width. Figs. 1 and 2, on Plate II., show the design of extractor mostly used in New Zealand.

## OTHER APPLIANCES.

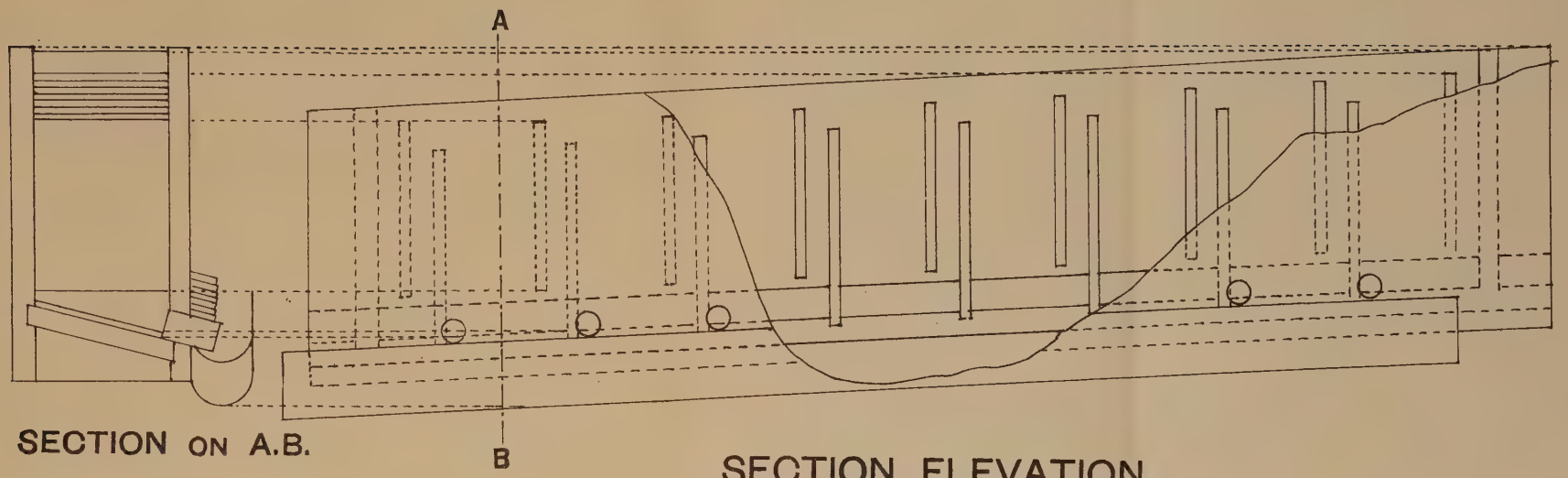
**The Roasting Furnace.**—In New Zealand cyanide plants this consists of a shallow cast-iron pan or plate, built over a small furnace. A funnel-shaped hood of light wrought-iron is placed over the furnace to carry off the zinc oxide and other fumes.

The hood is suspended by a chain or steel-wire rope passing over a pulley overhead, and is balanced by a weight on the end of the chain. The upper or narrow end of the hood telescopes for a few feet into the pipe leading to the condensing flue, and by means of the balance-weight the hood can be lowered or raised over the roasting-pan or plate as required.

The first part of the condensing flue is a nearly horizontal length of iron pipe, on which a stream of cold water is sometimes allowed to play, so as to condense the mercury vapours and zinc fumes.

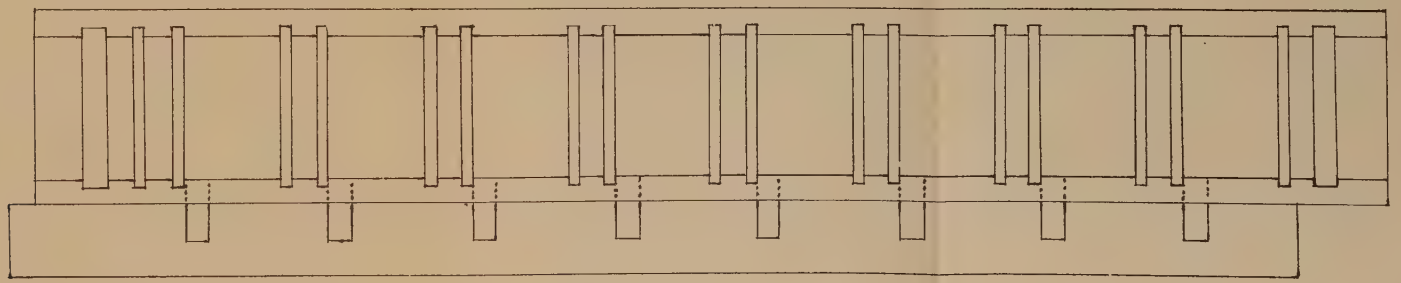
# ZINC EXTRACTOR.

*Plate II.*




SECTION ON A.B.

SECTION ELEVATION.



PLAN.

Scale of Inches

At the Witwatersrand goldfields a built-in, muffle-like roasting furnace is sometimes used, but it is doubtful if it possesses any advantages over the open roasting-pan.

A roasting furnace capable of taking 50lbs. of precipitate at a time is shown in Plate III., Figs. 1 and 2.

Fig. 3 show a special furnace for roasting slimes. It is a combined reverberatory and muffle-furnace. There is a fire-box at one end, a cast-iron hearth in the middle, on which the material lies, and an exit-flue at the other end. The fire-bridge is built in two parts, with a passage between them, leading to flues below the cast-iron plate, so that part of the heat is deflected and passes under the plate, while the remainder passes over it with a plentiful supply of air, thus effecting the oxidation of nearly all the zinc.

**The Filter-Press.**—In large works, where hundreds of pounds of precipitates have to be handled in the course of a week, a filter-press with a strong close canvas webbing is found very convenient for washing and drying. The false bottom of the press is connected with the air-pump.

**Tell-tales.**—These should be provided for the dissolving and solution tanks, and the storage sumps. They should be marked in feet and inches, so that the workmen can draw off a given depth of solution without any calculation. The cubic content of an inch of depth of each vat or tank in pounds should be marked on a board, so that when so many pounds of solution are required, their equivalent in inches of depth can be ascertained at a glance, thus saving delay and error from needless calculations.

## SIZE OF PLANT.

The size of the plant will depend on the intended capacity per month and the condition of the material to be treated. A plant capable of treating 2000 tons of freely

percolating tailings per month would probably not be able to treat more than half that quantity of dry-crushed ore or slimy tailings.

With dry-crushed ores the depth of the charge seldom exceeds 3ft., while with sharp tailings the depth sometimes reaches 13ft. or 14ft. Cyanide plants for the treatment of dry-crushed ore are therefore provided with a large number of shallow vats; and those for tailings with a less number of deep vats.

Below is given the number of leaching vats required for the treatment of different quantities of dry-crushed ore and tailings.

For Dry Ore or Slimy Tailings.—Size of vat: 22½ft. diameter, 4ft. deep; charge 30 tons.

| | | |
|---|---|---|
| 2 vats | .. .. | 350 tons per month |
| 4 ,, | | 700 ,, .. |
| 6 ,, | .. .. | 1,050 .. .. |
| 8 .. | .. .. | 1,400 ,, .. |
| 10 .. | .. .. | 1,700 ,, ,, |
| 12 ,, | .. .. | 2,000 ,, ,, |

For Freely Percolating Tailings.—Charge 100 tons.

| | | |
|---|---|---|
| 2 vats | .. .. | 1,400 tons per month |
| 4 ,, | .. .. | 2,800 .. .. |
| 6 ,, | .. .. | 4,200 .. .. |
| 8 ,, | .. .. | 5,600 ,, ,, |

In small plants it is advisable to have a spare leaching tank in case of a temporary breakdown in one of the tanks in use.

## COST OF PLANT.

The cost of the cyanide plant will depend largely on the locality, the material used in the construction, and to a certain extent on the condition of the material to be treated.

Fig. 1.

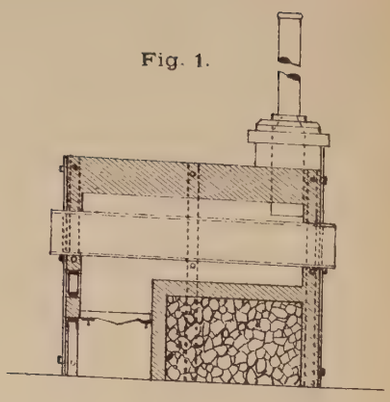
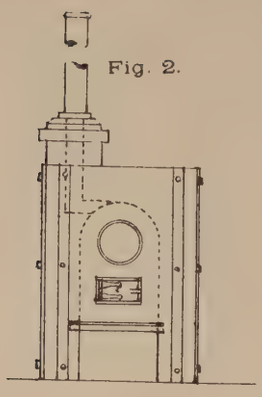


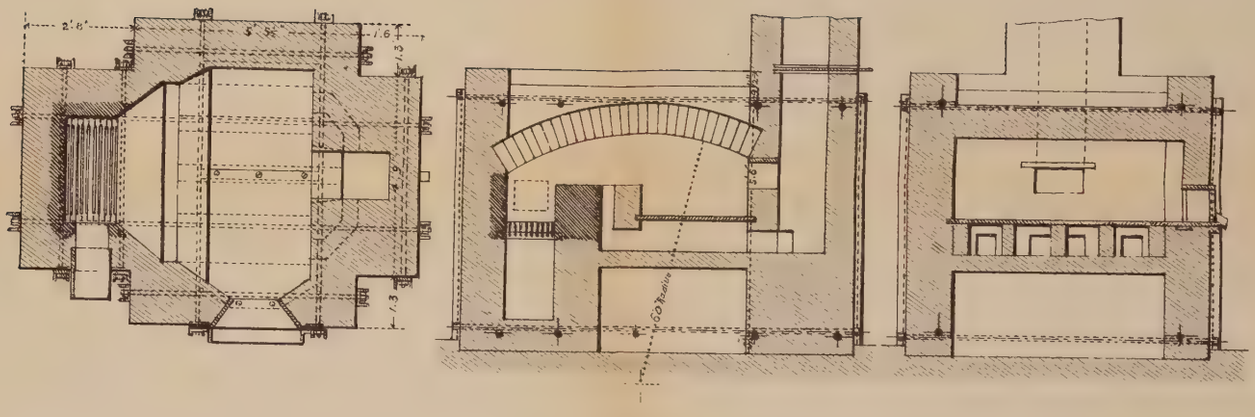
Fig. 2.



SCALE OF FEET

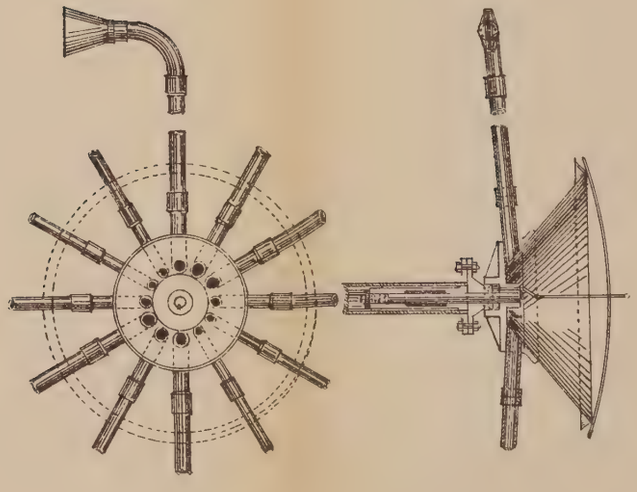
INCHES 12 8 4 0 1 2 3 FEET

Figs. 3



SCALE OF FEET.

INCHES 12 6 3 0 1 2 3 4 5 6 7 8 9 10 FEET



Figs. 4.

SCALE OF FEET.

INCHES 12 6 3 0 1 2 3 4 5 FEET

Wilsons & Horton Lith Auckland

Plate III.

For example, a plant to treat 2,000 tons of dry-crushed ore per month will cost more than a plant to treat 2,000 tons of tailings.

The approximate cost of plants of different capacities in New Zealand, when wood is the material used in the construction of the vats, foundations, and buildings, is given below :—

For Dry-crushed Ore—

| Monthly Capacity. | Cost. |
|---|---|
| 350 tons .. .. | £750 |
| 700 ,. .. .. | £950 |
| 1,050 ,. .. .. | £1,200 |
| 1,400 .. .. .. | £1,500 |
| 1,700 . .. .. | £1,800 |
| 2,000 ,. .. .. | £2,000 |

For Tailings—

| Monthly Capacity. | Cost. |
|---|---|
| 1,400 tons .. .. | £1,200 |
| 2,800 ,, .. .. | £2,000 |
| 4,200 ,. .. .. | £3,000 |
| 5,600 ,, .. .. | £4,000 |

The above estimates include the cost of the laboratory, assay and melting furnaces, and all the appliances for a successful cyanide plant.

The cost of steel or iron vats is about the same as that of wood. The steel leaching vats at the Waihi-Silverton, 16ft. diameter and 4ft. deep, with central discharge, cost £56 each, and the foundations of wood about £10 each.

At Johannesburg the cost of a cyanide plant is about 25s. per ton of tailings to be treated per month. Thus a plant to treat 3,000 tons of tailings per month would cost about £4,000. For larger plants the cost is less.

## CHAPTER VI.

### THE ACTUAL EXTRACTION BY CYANIDE.

**Synopsis of the Process.**—No hard and fast rules can be followed in the working of the cyanide process, as modifications have to be introduced to meet the special requirements of the different classes of ore; but the essential features of the process, whether for the treatment of ores or tailings, are practically the same in all cyanide plants.

The first operation is the filling of the vats. In the direct treatment of dry-crushed ore, the pulverized material is filled in to a depth of three or more feet, according to its fineness. With a uniform granular product, five or even six feet of ore may be filled into the vat, and, in the case of freely-percolating tailings, the vats may be safely charged to a depth of from six to ten feet.

With tailings it is the practice to fill the vat within a few inches of the top, because the strong solution or preliminary wash-water when applied causes the charge to shrink several inches.

In the case of acid ores or tailings, a preliminary treatment with a caustic alkali, or at least a water-wash, is necessary in order to remove, or neutralize, the mineral salts and acids which decompose the cyanide solutions.

The strong solution, containing generally from 0·3 per cent. to 0·8 per cent. of potassium cyanide, is then applied. The proper strength of solution must be determined experimentally, and with most ores it will be found advisable to

use a larger bulk of solution rather than unduly increase the strength. Silver ores require a larger bulk of solution than gold ores to obtain a fair extraction.

In the case of a dry-crushed ore the solution is allowed to permeate the mass from below, but in the case of tailings it is allowed to flow in over the surface, and soak downwards. The filling of the strong solution generally takes from two to four hours. With tailings it is allowed to stand in contact for 12 hours, after which percolation is begun. With dry-crushed ores slow artificial percolation is commenced at once.

The leaching with the strong solution takes from 24 to 36 hours, according to the character of the bullion; but in the case of ores containing a proportion of fairly coarse gold, it is customary to make the strong solution up to the working strength and pass it through the charge till an adequate extraction is obtained.

The weak cyanide solution, often called the *first wash*, is then applied. It is pumped from the strong sump, and generally varies in strength from 0·1 per cent. to 0·25 per cent. of potassium cyanide. It is allowed to percolate as rapidly as possible, the filtration being facilitated, when necessary, by means of an artificial vacuum, which may be created by a steam exhaust, or an air-pump connected with a vacuum cylinder.

The first weak cyanide wash is succeeded by two or three washes of solution from the weak sump, containing from 0·02 per cent. to 0·1 per cent. of KCy.

A final washing of clean water is then applied, which serves to displace the preceding weak cyanide wash. By this means the quantity of cyanide solution in circulation remains much about the same.

The quantity of the strong solution used is about one-third of the weight of the ore. The cyanide and water-washes are each about one-sixth of the weight of the ore.

During the treatment of slimy tailings it is found advantageous to sometimes turn the material over by hand-labour, and thereby effect a more complete washing out of the cyanide solutions from the slimes, which always have a tendency to entangle and retain them.

At the Witwatersrand Goldfields, in South Africa, the tailings are sometimes subjected to a double treatment—*i.e.*, after the first cyanide treatment the residues are charged into other leaching vats, and again treated with cyanide.

The average extraction there amounts to about 70 per cent., and with double treatment this is raised to 85 per cent. With tailings of fair value the extra recovery is said to leave a margin of profit.

The sequence in which the different operations are applied may be tabulated as follows :—

1. Filling the leaching vats.
2. The preliminary treatment with water or alkaline washes if necessary.
3. Leaching with the strong solution containing from 0·3 per cent. to 0·8 per cent. of KCy.
4. First wash, with cyanide from strong sump, containing from 0·1 per cent. to 0.25 per cent. KCy.
5. Second wash, with cyanide from weak sump, containing from 0·02 per cent. to 0·1 per cent. KCy.
6. Third wash, same as second.
7. Fourth wash, same as second and third.
8. Fifth wash, with clean water.

The gold and silver in the ore are dissolved by the strong solution, and removed or carried out by the first and second washes. The cyanide solutions are allowed to flow through the zinc-extractors, the strong solutions through the strong box, and the weak solutions through the weak box. The first two washes, which generally carry most of the dissolved bullion, are conducted through the strong extractor.

FIRST STAGE.

**Filling the Vats with Dry-Crushed Ore.**—In the case of dry-crushed ores, the charging of the vats is a simple operation, the only disadvantage being the clouds of dust, which seem to be inseparable from the handling of dry-pulverized ores. The vats are generally charged by trucks running on to a traveller, provided with hand-traversing gear so as to enable the sand to be tipped in different parts of the vat. In order to prevent packing, the sand is discharged from the trucks on to the platform of a small traveller fixed below the main traveller. By this means the pulverized material is dispersed in a gentle shower over the whole vat.

At the Kapai-Vermont Cyanide Works, Kuaotunu, N.Z., the vats are filled directly from the dust-bin, which is situated overhead in an elevated position. By means of a chute, provided with a universal canvas joint, the material is evenly spread over the vats.

When the vat is charged the surface of the ore is made smooth by passing a wide shallow wooden hoe or rake over it.

**Filling the Vats with Tailings.**—One of the most serious disadvantages of wet-crushing with the stamp-mill is the production of fine slimes. All ores, even the most

silicious, form a proportion of slimes when wet-stamped; and when clayey or earthy matter, country-rock, iron or manganese oxides are associated with the ore, the proportion of slimes is often excessive.

In many places the slimes are very valuable, sometimes even more so than the sands. In leaching processes they are very prejudicial to the treatment, as they seriously interfere with the percolation and washing, thereby causing the extraction to be both imperfect and costly. When the slimes are irregularly distributed through the sands, the cyanide solutions form channels through them, and imperfect leaching is the natural result.

At the Witwatersrand Goldfields, where wet-crushing, followed by copper-plate amalgamation, is at present universal, there are several methods in use for dealing with the tailings before treatment with cyanide. At the extensive works of the Langlaagte Estate Gold Mining Company, near Johannesburg, the tailings, after leaving the plates, are concentrated, and then run into three large settling-dams, each capable of holding 7,000 tons. The sands settle in the dam while the slimes are carried off by the overflow, and allowed to run away. The tailings, now free from slimes, are hauled from the dams in trucks by means of two endless steel wire ropes, and run up to an overhead tram-line, from which they are dumped into the leaching vats ready for treatment.

At the Robinson, Princess, and many other mines at Johannesburg, the tailings, after being subjected to concentration, are run into intermediate settling-tanks, which are circular in shape and sufficiently large and numerous for the requirements of the mill. At the Robinson plant, treating 330 tons per day, there are six circular wooden

vats, each 24ft. in diameter, and 11ft. in depth. This gives a settling surface of 32 square feet per stamp.

From these intermediate settling vats the tailings are distributed to the leaching vats. When the slope of the ground permits it, the settling vats are placed above the leaching vats, the tailings being discharged from holes in the bottom; but when they are below the level of the leaching vats, the tailings are hauled up in trucks actuated by endless wire ropes.

To ensure a fairly even distribution of the sand and slimes in the leaching vats, Messrs. Butters and Mein have invented a simple and ingenious automatic distributor, which is now in use at the Witwatersrand. It consists of a central casting, with a vertical spindle revolving in a foot-step, and carrying a conical hopper at the top, from which radiate 12 or 16 iron pipes with bent ends. These pipes vary in size from 1½in. to 2½in. in diameter. To their discharge ends are attached flattened nozzles to assist in spreading the tailings over a wide area. A coarse screen is placed over the central hopper, or bowl, so as to prevent stones or pieces of wood, or other debris, from choking the pipes. The distributor is fixed on an iron column in the centre of the vat, and the reaction of the pulp escaping from the bent pipes causes it to slowly revolve in a manner similar to that of an ordinary garden sprinkler. (Figs. 4, Plate III.)

To ensure success, the settling vat must be filled with clean water before admitting the pulp, otherwise the slimes would settle with the sand until the overflow of water began. While the machine is running, there must be a continual overflow from the vat to carry off the fine slimes. The discharge, or overflow, from the settling vat takes place

at all points of the circumference, being received into an annular ring, surrounding the top of the vat, and which conveys it to the slime-pit.

The vats are provided with filter-beds, and when full of tailings, the water is allowed to drain off for 15 or 24 hours.

**Direct Filling.**—This method is in use at the works of the City and Suburban, Crown Reef, and other companies at the Witwatersrand, and consists in conducting the pulp, as it leaves the copper-plates, into a classifier or spitzlutte. In this the pulp is divided into two streams: one, the overflow, carrying slimes and fine sands; the other, carrying the coarse sands, together with some fine sand and slimes, which are conveyed by an india-rubber hose to the leaching vat, where they are distributed by moving the hose over the whole area of the vat. The excess of water is carried off by adjustable gates, fitted inside the vat, taking with it some fine sand and slimes.

The advantages of direct filling are :—

1. That pyritic tailings are exposed to the minimum of oxidation.
2. A second handling of the tailings is avoided.
3. A rough preliminary classification is effected, thus separating the fine slimes.

The principal disadvantages of this method are : —

1. The packing of the tailings, which prevents a complete draining of the contained water.
2. The unequal distribution of the sands and slimes, which militates against a perfect extraction by favouring the formation of channels in the mass during the subsequent leaching.

The method of classifying the pulp by a spitzlutte, and direct filling, have been in operation at the Great Mercury Cyanide Works, Kuaotunu, N.Z., since 1892. When the leaching vat is charged, the water is drawn off through the filter-bed, and after the tailings have drained they are turned over by hand so as to loosen them, and at the same time thoroughly mix any slimes present with the sands.

SECOND STAGE.

**The Preliminary Water or Alkaline Wash.**—This treatment is only necessary in the case of ores or tailings containing the decomposition products of pyrites, or other base sulphides.

The drying of a pyritic ore in a kiln for dry-crushing and direct treatment generally results in the formation of soluble sulphates, which are destructive to cyanide. At the works of the Woodstock Gold Mining Company, at Karangahake, N.Z., the consumption of cyanide from this cause was very heavy; but the preliminary treatment with a caustic alkali, according to the recent report of the management, has effected a great saving of cyanide, accompanied by a higher extraction.

The products of the partial oxidation of iron pyrites in tailings are principally free sulphuric acid, soluble sulphates and insoluble basic sulphates, all of which cause a large consumption of cyanide, and must, therefore, be removed or neutralized before cyanide treatment is commenced.

If the tailings are very acid, and a considerable proportion of the salts are found to be soluble in water, the practice in New Zealand is to apply a preliminary water-wash. In order to neutralize the remaining acid, a quantity of a solution of caustic soda, equal to about 1lb. of the salt to every ton of ore, is allowed to stand in contact with the

tailings for two hours and then drained off into the alkali-sump.

The necessary quantity of caustic soda can be determined experimentally by the method recommended by Feldtmann. (See page 41.)

At the Witwatersrand Goldfields powdered caustic-lime is generally used in place of soda. With very acid tailings as much as 2½lbs. to the ton is used. It is applied by adding the requisite amount to each truck-load before being dumped into the leaching vats.

The author has generally used lime at the Government Experimental Works at the Thames, and found it preferable to caustic soda, as it is not attended with the production of ferrocyanide of zinc, which fouls the zinc in the extractor.

In the case of very acid tailings, Feldtmann strongly condemns the practice of conducting the preliminary washing in the leaching vat on account of the possibility of the acid acting on the residual cyanide in the vat, and thereby liberating sufficient hydrocyanic acid to dissolve an appreciable quantity of gold, which would be lost, as it is not precipitated by zinc.

He suggests that the washing should be effected in one vat, and the leaching in another; and considers that the extra cost of handling would be more than made up by the higher extraction.

THIRD STAGE.

**Strong Solution Leaching.**— With dry-crushed ore, the strong solution, about *one-third* the weight of the charge, and generally containing from 0·3 per cent. to 0·8 per cent. of available potassium cyanide, is allowed to pass into the vat from below. When the solution stands *two inches* above the surface of the ore the stop-cock is shut, and

the solution lying below the filter is then drawn off, and filtration commenced. The strong solution generally takes from 24 to 36 hours to percolate through.

In the case of tailings, the strong solution is added on top. After standing 12 hours to allow the solution to permeate all the lumps of slimy material, the solution is slowly drained off, and passed through the zinc-extractor.

When the ore contains a large proportion of silver—say, from five to eight parts to every part of gold—it will be found necessary to adopt one of two courses in order to obtain a satisfactory extraction.

Either a much greater bulk of potassium cyanide solution must be used to leach the ore—say, a quantity equal in weight to that of the ore taken—or else a much stronger solution must be used.

The former, only, of these two courses would be applicable if the ore contained even a small proportion of copper oxide, carbonate, or sulphide, or the sulphide of antimony, since the solubility of these in all solutions of cyanide, but especially in the strong, would render a fair extraction impossible, besides causing a heavy loss of cyanogen.

The sulphide ores of silver are more slowly soluble in cyanide than gold, and for this reason the treatment of such is always more expensive than that of ordinary tailings or gold-bearing ores. On the other hand, the chloride of silver (kerate or horn silver) is more readily acted on than gold, the extraction generally exceeding 80 per cent.

In the practical treatment of ores and tailings by cyanide, one of the first anomalies to attract the notice of the metallurgist is the fact that the strong solution, while it loosens the gold, so to speak, yet does not carry it away, this being effected by the first and second cyanide washes.

It is found that the first portions of the strong solution, draining from the charge, contain only from 0·02 per cent. to 0·1 per cent. of cyanide; but the remaining portions come off stronger and stronger after a lapse of eight or twelve hours, until, towards the end of the strong leaching, the maximum strength is reached, after which the strength declines a little before the application of the first cyanide wash.

The first portions of the solution during the strong cyanide leaching are, therefore, being weak in cyanide and low in gold, passed through the weak zinc-extractor, while the later portions, together with the first and second cyanide washes, are conducted to the strong extractor.

**Strong Sump-Solution Wash.**—After the strong solution has completely drained off, the strong sump-solution is applied from above, being run in on the top, or surface, of the ore. Its strength varies from 0·1 per cent. to 0·25 per cent. of KCy, and the quantity applied from one-third to one-quarter of the weight of the ore.

This weak solution is sometimes allowed to slowly percolate through the charge, but more often it is drawn off as rapidly as possible, for the more rapidly the wash solutions are drawn off the more effective is the washing. The filtration is assisted by opening the stop-cock connecting the bottom of the leaching vat with the vacuum-cylinder.

The percolation of the weak solution generally takes from 12 to 20 hours, the time depending on the condition of the pulp forming the charge.

**Weak Cyanide and Water-Washes.**—The number and strength of these will depend entirely on the character of the ore, or tailings, being operated on. In

some cases it is found necessary to apply as many as three or four weak cyanide washes, from the weak sump, and then a final water-wash; in other cases the whole treatment consists of the strong cyanide leaching followed by a cyanide wash from the strong sump, a cyanide wash from the weak sump, and one or two final water-washes.

The quantity of each wash is in most cases one-half that of the strong cyanide solution.

The effect of the different washes should be carefully determined by assaying the residues after each washing, and also the wash solutions as they drain from the vat. By this means the necessary number of washings will soon be ascertained.

The assay sample should represent a fair average of the charge in the vat, and is easily and reliably obtained by taking a large number of cores, the full depth of the charge, by means of a tube shaped something like a cheese sampler. The cores should be dried and then sampled down for assay.

The different operations to be undertaken in the dry-crushing and direct method of treatment may be summarized as follows:—

1. Dry-crushing.
2. Cyanide treatment.
3. Copper-plate amalgamation of tailings.
4. Concentration of tailings from plates.
5. Treatment of concentrates.

With wet-crushing the different operations are:—

1. Wet-crushing.
2. Copper-plate amalgamation.
3. Concentration.

4. Treatment of concentrates.
5. Treatment of tailings by cyanide.
6. Treatment of slimes by cyanide.

## THE TREATMENT OF SLIMES.

With all methods of wet-crushing and pulverizing, the formation of a certain proportion of slimes seems inevitable, and when the ore contains metallic oxides, clay, or other earthy matter, the production is often very large. In ores containing a certain proportion of very fine or "float" gold, the slimes are generally a valuable product, and their successful treatment has engaged the attention of metallurgists for many years.

Since the introduction of the cyanide process many attempts have been made to leach the slimes resulting from wet-crushing, on an economic scale, but, so far, no satisfactory process has yet been offered to the public. The problem is principally a mechanical one, consisting in the difficulty of separating the solutions from the slimy mass, rapidly, effectively, and at such a cost as to permit the treatment of slimes of low value.

Up to the present a great many different devices have been tried with varying degrees of success. Among these may be mentioned compression by hydraulic and other presses; agitation and centrifugal force; agitation and decantation; and agitation with filtration, aided by an artificial vacuum.

At the Try Fluke mine at Kuaotunu, N.Z., the slimes are dried in the sun, broken up, and then mixed with the coarser tailings in the proportion of one part of slimes to two parts of tailings. This method gives very satisfactory results; but it cannot be carried on in the winter or wet season, and at all times the cost of labour is too great to

permit its adoption for the treatment of low-grade slimes, such as those valued at from 8s. to 12s. per ton.

The dry-crushing of ores by means of the Californian stamper-battery is always attended with the production of so large a proportion of slimes that only a shallow depth of the pulverized material can be leached in the direct method of cyanide treatment.

In New Zealand the depth of dry pulverized material placed in the leaching vats seldom exceeds three feet, even with the most favourable silicious ores. This necessitates a large plant to treat a comparatively small output of ore, and a correspondingly greater cost per ton for treatment.

Early in 1893, the author treated a large parcel of ore from the Monowai mine, in the Thames District. The ore consisted of hard bluish, and grey-coloured, splintery quartz, containing a considerable proportion of sulphides of iron, copper, lead, and zinc.

The ore was dried, dry crushed, sampled and assayed, showing a value of £5 5s. per ton. The crushing was effected in a stamper-battery, which produced a large quantity of the finest slimes. These slimes rendered it impossible to effect the leaching by percolation, even with the aid of a vacuum. When mixed with water, the slimes, when only 4in. thick, settled on the filter-cloth, forming an impervious bed, through which it was found impossible to draw off the solution

The pulp was then subjected to agitation, by which the dissolution of bullion was effected in six or seven hours. The separation of the solution from the pulp, however, was a long and tedious operation, and extended over eight days. It was effected, but not very satisfactorily, by agitating the ore, allowing the slimes to settle, and then drawing the

clear solution off by a syphon. The weak solution and wash-waters were added in succession, and the same operation performed after each.

In order to ascertain the degree of fineness to which the ore was reduced when crushed, a number of experiments were made with a 60-mesh, 40-mesh, and ordinary battery-punched screen, and found that the results were in each case, as follow:—

With 60-mesh—

| | | | | |
|---|---|---|---|---|
| A. | 18 % | remained on .. | 8,100 | sieve |
| B. | 5 % | ,, ,, | 3,600 | ,, |
| C. | 100 % | passed through a | 1,600 | ,, |

With 40-mesh—

| | | | | |
|---|---|---|---|---|
| A. | 22 % | remained on .. | 8,100 | sieve |
| B. | 12 % | ,, ,, | 3,600 | ,, |
| C. | 3 % | ,, ,, | 1,600 | ,, |
| D. | 100 % | passed through a | 900 | ,, |

With Punched-Screen, equal 30-mesh—

| | | | | |
|---|---|---|---|---|
| A. | 26 % | remained on .. | 8,100 | sieve |
| B. | 18 % | ,, ,, | 3,600 | ,, |
| C. | 13 % | ,, ,, | 1,600 | ,, |
| D. | 4 % | ,, ,, | 900 | ,, |
| E. | 100 % | passed through a | 625 | ,, |

Taking the values of the different products separately, it was found that the finest in all cases gave the highest values. This also received confirmation from the circumstance that the fine dust, which had collected on an elevated platform during crushing, assayed higher than the dry material in the bin.

The relative values were as follow:—

| | | |
|---|---|---|
| From dust-bin | .. .. | £5 5 0 per ton |
| ,, platform | .. .. | 6 13 6 ,, |

The ratio of gold to silver in the pulp from the dust-bin was nearly 1 to 9, and in the dust from platform 1 to 12, thus showing an increase of the silver contents.

Acting on a suggestion thrown out by Mr. C. Wichmann, I obtained from Mr. E. G. Banks, metallurgist to the Waihi Gold Mining Company, the results of a number of experiments, showing the relative values of the different degrees of fineness of the pulverized ore. Mr. Banks has had charge of the extensive cyanide works of the Waihi Company since their inception in 1893, and has made a large number of interesting and valuable experiments in connection with his department. The results of his experiments are very instructive:—

EXPERIMENTS *showing degree of fineness and relative values to which "Martha" ore was reduced by stamping through 30-mesh screens.*

| | | Value per ton. |
|---|---|---|
| A. | 0·36 % remained on 30-mesh (900 holes) .. | £4 2 6 |
| B. | 2·16 % ,, ,, 40-mesh (1,600 holes).. | 3 18 10 |
| C. | 9·29 % ,, ,, 60-mesh (3,600 holes).. | 3 11 2 |
| D. | 25·72 % ,, ,, 80-mesh (6,400 holes).. | 4 9 0 |
| E. | 74·28 % passed ,, ,, .. | 5 7 1 |

*Similar Experiments with 40-mesh Screens.*

| | | |
|---|---|---|
| A. | 0·3 % remained on 40-mesh (1,600 holes) .. | 3 12 7 |
| B. | 7·8 % ,, ,, 60-mesh (3,600 holes) .. | 3 7 11 |
| C. | 14·7 % ,, ,, 80-mesh (6,400 holes) .. | 3 11 4 |
| D. | 85·3 % passed ,, ,, .. | 3 19 2 |

*Experiments showing value of Dust rising from stamps—Dry-crushing.*

| | | Value per Ton. |
|---|---|---|
| A. | Dust obtained from floor of mill to 10ft. high | £6 16 4 |
| B. | ,, ,, 10ft. to 20ft. high .. | 7 2 4 |
| C. | ,, ,, 20ft. to 30ft. ,, .. | 7 3 1 |
| D. | ,, ,, 30ft. to 40ft. ,, .. | 7 19 11 |
| E. | ,, ,, 40ft. to 60ft. ,, .. | 8 13 8 |
| Average value of ore from which this dust came | | 4 1 4 |

Cyanide Experiments on "Martha" ore, crushed by Krom Rolls.

Fineness .. All passed a 40-mesh screen
32 % remained on a 60-mesh ,,
48·4 % ,, ,, 80-mesh ,,

(A.) 2,000*grs. of average sample, agitated with a* ·33 % *solution of KCy for* 24 *hours.*

| | Gold. | Silver. | Saved % Gold. | Saved % Silver. |
|---|---|---|---|---|
| Assay before treatment | 0 14 20 | 3 9 6 | | |
| ,, after ,, | 0 3 2 | 2 3 0 | 79·2 | 37·9 |

(B.) 2,000*grs. ground to equal stamps on* 40-*mesh, agitated as in A.*

| | | | | |
|---|---|---|---|---|
| Assay before treatment | 0 14 20 | 3 9 6 | | |
| ,, after ,, | 0 1 12 | 1 15 5 | 89·9 | 49·2 |

(C.) 2,000*grs. of what would not pass* 60-*mesh, agitated as in A.*

| | | | | |
|---|---|---|---|---|
| Assay before treatment | 0 12 6 | 2 13 2 | | |
| ,, after ,, | 0 5 2 | 2 2 19 | 58·5 | 19·4 |

(D.) 2,000*grs. of what would not pass* 60 *and* 80-*meshes, agitated as in A.*

| | | | | |
|---|---|---|---|---|
| Assay before treatment | 0 12 15 | 2 16 17 | | |
| ,, after ,, | 0 3 20 | 1 18 0 | 69·6 | 32 9 |

(E.) 2,000*grs. of what passed an* 80*-mesh screen, agitated as in A.*

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Assay before treatment | 0 | 15 | 17 | 3 | 13 | 5 | | |
| ,, after ,, | 0 | 1 | 12 | 1 | 15 | 0 | 90·5 | 52·2 |

The results of these experiments with material pulverized both with stamps and rolls showed conclusively that the finest products contained the highest values, and that the highest extractions were obtained from the finest sands.

For some years the author has devoted much thought and research to the slime problem, having conducted a large number of experiments at the Experimental Plant at the Thames School of Mines, to discover a satisfactory and practicable process. In 1893 he obtained, with two of his senior students, a caveat for a combined agitation and leaching process, which may be described as follows :—

The appliances used in the operation consist of a shallow circular vat, a vacuum-cylinder, and an air-pump. The vat is provided with four revolving arms, to which soft rubber brushes are attached. The bottom of the vat is provided with a false-bottom, consisting of a wooden grating covered with wool packing or other webbing. The operation is conducted as follows :—The leaching solution, made up to the required strength, is first conducted into the vat. The revolving arms are then set in motion, and the dry pulp or fine slimes introduced. The agitation is continued for six hours, or until the extraction is complete. A stop-cock, in a pipe connecting the false-bottom of the vat and the vacuum-cylinder, is then opened, and the air-pump started. The effect is immediate. At once the clear solution begins to drain over into the vacuum-cylinder, the brushes on the revolving arms preventing the slimes from settling and choking up the filter-cloth. When the slimes have been drained down to a thick paste, the first wash is added,

the pump again started, and the slimes drained as before. The subsequent washes are applied in the same manner, and, when the washing is completed, a plug or door is opened, and the leached slimes are sluiced out. The whole operation of leaching and washing takes from 18 to 24 hours.

This process was adopted by the author for the treatment of several tons of ore from the Monowai mine, at Waiomo, with complete success. This ore contained a large proportion of clay and iron oxides, and, when dry-pulverized, formed a pulp which defied all the ordinary methods of percolation. Trial tests were also made with parcels of very fine slimes, and in all cases the results were most satisfactory; but experience showed that the amount of motive-power and the limited size of the charges rendered the cost, per ton, too great for the adoption of the process for the treatment of low-grade slimes on a commercial scale.

The author continued his experiments for the treatment of slimes in another direction, and in 1895, with Mr. G. W. Horn, a former student, obtained a caveat in New Zealand for an improved leaching process for the treatment of slimes and other fine matters by cyanide or other solvents.

The necessary apparatus consists of a solution vat, an air-compressor, and a leaching vat provided with a filter-frame and webbing, both at the top and bottom. The vat may also be provided with an airtight cover, so as to permit an artificial vacuum to be used to facilitate the filtration.

In practice the leaching vat, which may be constructed of wood, or any suitable material, and of any convenient size, is charged with cyanide solution. The slimes are then

introduced and well mixed with the solution, which is then allowed to penetrate the slimes and stand in contact with them for two hours. At the end of that time, the top filter-frame and webbing is put on, and additional cyanide, or other solution, is allowed to penetrate and permeate the slimes from below, thus displacing the solution already in the vat, causing it to pass through the top filter-web as a clear liquid, which is conducted away by a launder.

In order to prevent the solution selecting channels through the slimes, or other matter in the vat, during the upward filtration, the material is agitated by the discharge of compressed air through it from distributors placed in the sides and bottom of the vat.

The gold in the slimes being necessarily very fine, is dissolved very quickly, and the percolation can generally be started two hours after charging the leaching vat. When a portion of the gold is coarser than usual, or when the bullion exists in the form of amalgam, it is found that the agitation caused by the discharge of compressed air during the leaching greatly accelerates the dissolution.

The novel features of the process consist of upward filtration, and agitation by means of discharges of compressed air through the mass.

The results of a number of small tests, on the finest slimes obtainable, have proved most satisfactory, and more extensive trials will soon be undertaken. One air-compressor would be sufficient for a number of leaching vats; and the author is confident that this process is capable of very wide and useful application.

## TREATMENT OF CONCENTRATES.

Pyritic concentrates may be leached *by agitation* with cyanide, or simply *by percolation*, as with ordinary tailings. Both methods give satisfactory results.

Leaching by Percolation.—At the Witwatersrand Goldfields the treatment of concentrates by cyanide has been largely adopted in preference to chlorination.

From the storage vats the concentrated material is taken to the leaching tanks, and subjected to the action of strong cyanide solution, containing from 0·4 per cent. to 0·6 per cent. of cyanide, for periods varying from 12 to 18 days. In practice the solution is allowed to slowly percolate through the concentrates, and is then passed through the zinc-precipitation boxes. It is again made up to the original strength and allowed to percolate as before. This operation is continued until a satisfactory extraction is obtained. At the Crown Reef Cyanide Works the cost of this method of treatment is said not to exceed 17s. per ton of 2,000lbs.

The treatment of ores and concentrates by agitation with cyanide is fully described in Chapter VIII.

## CHAPTER VII.

### THE APPLICATION OF THE PROCESS AT DIFFERENT CYANIDE WORKS.

**New Zealand.**—In the Waihi, Waitekauri, and Karangahake districts of the Hauraki Goldfields, the practice of dry-crushing and direct cyanide treatment is universal.

The ore of the celebrated Martha lode at the Waihi mine is typical of most of the ores being treated in these districts. It consists principally of hard, splintery, whitish-grey chalcedonic and crypto-crystalline quartz, often possessing a banded and wavy structure. It is perfectly free from all base metallic sulphides, and the amount of iron oxides present is so small that when roasted and pulverized the colour of the dust is perfectly grey.

The value varies from £4 to £5 per ton, the precious metals existing in the proportion of about 3oz. silver to 1oz. gold. The free gold is alloyed with about 35 per cent. of silver, being valued at about 53s. per ounce. The greater proportion of the silver exists in the form of the bluish-grey sub-sulphide known as argentite. In the surface levels, thin leafy plates of gold were not infrequently seen adhering to the surface of large cuboidal masses of quartz, but in the lower levels a *colour* is rarely seen, the gold existing in an extremely fine state of subdivision. Such an ore is theoretically perfect for cyanide treatment, and actual

experience has proved it to be so. By the old stamp battery, and copper-plate amalgamation, the recovery amounted to only some 4dwts. per ton, equal to about 15 per cent. of the value. By dry-crushing and pan-amalgamation the extraction was raised to 60 per cent., but at present the actual extraction by the cyanide process amounts to over 90 per cent. of the assay value.

The following particulars of the method of treatment at the Waihi works were supplied by Mr. H. P. Barry, the general manager, for the annual report of the New Zealand Mines Department for 1894:—

**Drying the Ore.**—The ore is trucked to the drying kilns, which consist of open circular holes excavated in the solid rock, their dimensions being 37ft. in depth and 20ft. in diameter at the top, and tapering somewhat at the bottom. Each kiln is capable of holding 100 tons of ore at a charge. The lower part is lined with bricks, and finished off with a brick arch, having a door and an iron chute for discharging the dried ore into trucks, which have access to the kiln by means of a tunnel cut in the rock.

**Charging the Kilns.**—The kilns are charged with alternate layers of wood and ore, the layers of wood being about 5ft. apart. When the kiln is fully charged, the wood is lighted, and, after it is all burned up, about half the charge, that is about 50 tons, is withdrawn and another 50 tons of raw ore, together with the necessary wood, are placed on the top. After this, about 50 tons of ore are withdrawn every third day, while a similar quantity of raw ore and wood is added.

This method of drying the ore is found to be very economical, as there is not a large surface of cold ore to

heat up as in the case of smaller kilns, which are completely emptied when the charge is dry. One ton of wood will dry about three tons of ore. The cost of firewood at the Waihi big mill is 1s. 7d. per ton of ore dried, and the total cost of drying, including labour, is 2s. per ton.

**Crushing and Pulverizing.**—From the kilns, the dried ore is trucked to the rock-breakers, whence it passes by gravitation to the self ore-feeders. The pulverizing machinery consists of a 90-stamp battery and an Otis ball-mill, having a capacity of about 10 stamps. The ore is passed through a 40-mesh screen.

**Filling the Cyanide Leaching Vats.**—From the screens, the dry dust falls into a long, narrow trough running parallel with the stamp-mortars, along which it is conveyed to the dust-bin at one end of the mill by means of an Archimedian screw. From the dust-bin the pulverized ore is lifted by a bucket-belt elevator and discharged on to an 8in. rubber-belt provided with rope edges, and by this conveyed to, and across, the dust-hopper, which is 110ft. long, running the entire length of the cyanide-plant house.

The dust-hopper has twenty doors for discharging the dust into the trucks, which are run straight out over the leaching vats on to travellers running on rails. The travellers are provided with hand-traversing gearing, thus enabling a truck to be tipped at any part of the vat. This is an important feature, as the finely pulverized material has a tendency to pack if moved about, or touched, in any way after being tipped into the vat.

As a further preventative against packing, there is a small traveller fixed below the main traveller, provided

with a platform at the height the ore has to be filled up to. All the trucks are tipped over this platform, which breaks the fall of the dust, and throws it in a light shower all around.

**The Cyanide Treatment.**—The following particulars of the cyanide treatment were kindly supplied to me by Mr. E. G. Banks, the chemist and metallurgist in charge of the cyanide operations. The plant consists of thirty circular leaching vats, each 22½ft. in diameter and 4ft. deep, together with the necessary dissolving and solution vats, sumps, extractors, vacuum cylinders, solution and air-pumps, etc.:—

| | HOURS. |
|---|---|
| Filling vat, 30 tons, two men .. .. .. | 2½ |
| Strong solution, 10 tons, 0·35 % KCy, leaching .. | 30 |
| Weak solution, 7 tons, 0·1 % KCy, with vacuum .. | 15 |
| First water wash, 6 tons, with vacuum .. .. | 24 |
| Second ,, ,, ,, .. .. | 36 |
| Discharging vat, one man sluicing .. .. | 2 |
| Taking up and cleaning filter-bottom .. .. | 4½ |
| Total .. .. .. .. | 114 |

A vacuum of 20in. to 23in. is maintained to obtain the above results. The average value of the ore is about £4 per ton, and the actual extraction from 90 % to 92 % of the original value, at a cost of 7s. 6d. per ton, not including royalty.

The particulars of the treatment of the tailings at the cyanide works of the Waihi Gold and Silver Mining Company are very instructive and interesting. The tailings are the residues resulting from the pan-amalgamation of the dry-crushed ore before the introduction of the cyanide process. The ore was crushed through a 60-mesh screen, and

pan-amalgamated in charges. The residues were discharged from the settlers into large dams, where they were allowed to settle. They consisted mostly of fine sands and a good deal of slimes. They contained no base metallic impurities, and the bullion existed principally in the form of amalgam.

Some 25,000 tons of these tailings were successfully treated by the Cassel Gold Extracting Company, whose works have recently been acquired by the Waihi Gold and Silver Mining Company, who are now treating the remainder of the tailings on their own account. The plant consists of eight leaching vats, each 22½ft. in diameter and 4ft. deep, together with all the necessary appliances.

The details of the cyanide treatment adopted for the treatment of these tailings are given below in tabulated form:—

CYANIDE TREATMENT OF WAIHI TAILINGS.

| | HOURS |
|---|---|
| Filling leaching vat, 30 tons, three men .. | 8 |
| Preliminary lime or water wash, 6 tons, with vacuum | 6 |
| LEACHING— | |
| Strong solution, 8 tons, 0·6 % KCy .. .. .. | 30 |
| Weak solution, 4 tons, 0·2 % KCy (from strong sump) .. .. .. .. .. | 12 |
| WASHING, USING VACUUM— | |
| First weak cyanide wash (from weak sump), 4 tons | 12 |
| Second ,, ,, ,, ,, ,, | 12 |
| Third ,, ,, ,, .. ,, | 12 |
| Fourth, water wash, 4 tons .. .. .. | 12 |
| Discharging vat, one man sluicing .. .. | 4 |
| Total .. .. .. .. .. | 108 |

Remarks.—The tailings are generally very clean, and the preliminary lime or water wash is not always applied. The strong solution is allowed to stand in contact with the tailings about four hours before the percolation is commenced. The average value of the tailings treated so far has been about 24s. per ton, and the actual extraction about 75 per cent., at a cost of 8s. per ton.

At the Golden Cross section of the Waitekauri Gold Mining Company's Special Claim, near Waihi, the ore is dry-crushed with stamps to pass through a 40-mesh screen and treated directly with cyanide. As a small proportion of the gold is coarse, the tailings are passed over amalgamated copper-plates, 30ft. long and 3ft. wide, set with a fall of 1 in 12.

The details of the cyanide treatment of this ore, for which I am indebted to Mr. R. Mellett, the metallurgist, are given in the following tabulated statement:—

| | HOURS. |
|---|---|
| Filling vats, three men, 22 tons .. .. .. | 3½ |
| Strong solution, 9 tons, 0·45 % KCy .. .. | 48 |
| Weak solution, 9 tons, 0·2 % ,, .. .. | 18 |
| First weak cyanide wash, 5·5 tons, 0·05 to 0·15 % .. | 18 |
| Second ,, ,, 5·5 ,, ,, .. | 18 |
| Third ,, ,, 5·5 ,, ,, .. | 18 |
| Fourth, water wash 3·0 ,, .. .. | 18 |
| Discharging vat, one man sluicing .. .. | 14½ |
| Total .. .. .. .. | 156 |

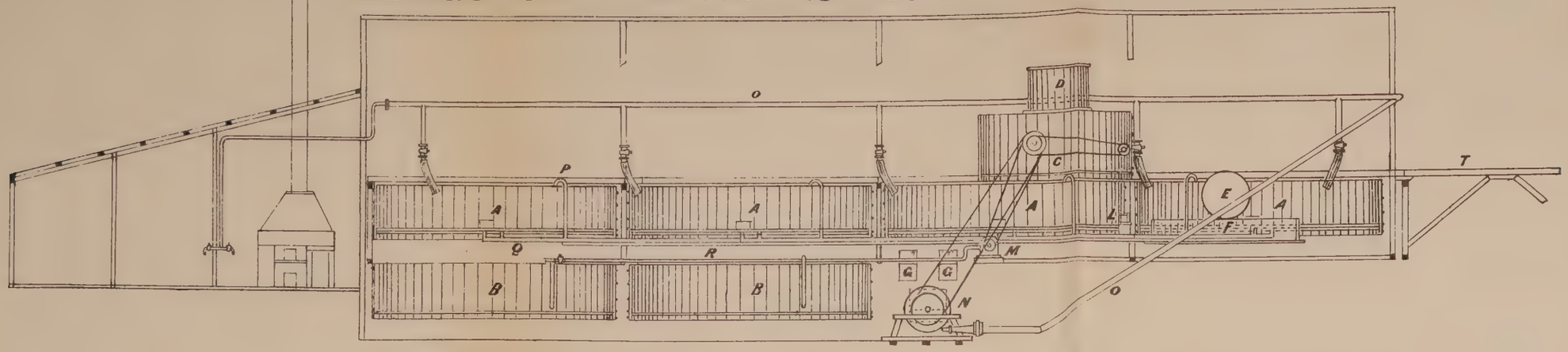
The average value of the ore treated during 1894 was £4 15s. per ton; and the actual extraction varied from 91 per cent. to 93 per cent., at a cost of 8s. 6d. per ton.

At the Crown and Woodstock Cyanide Works at Karangahake, amalgamated copper-plates are also used

# TAILINGS CYANIDE WORKS ERECTED BY THE CASSEL GOLD EXTRACTING CO., LTD.,

## AT WAIHI, N.Z.

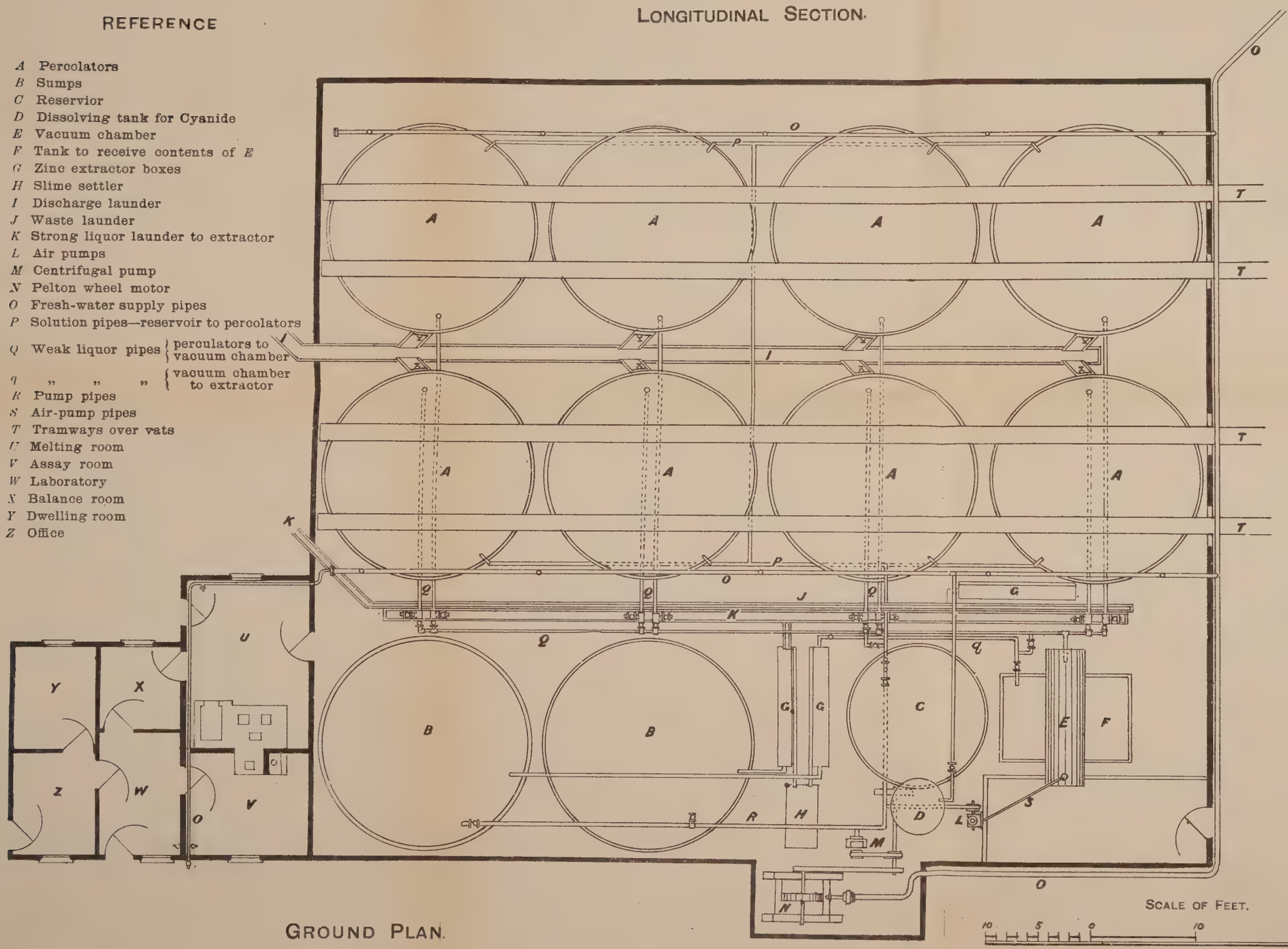
Plate IV.




LONGITUDINAL SECTION.

REFERENCE

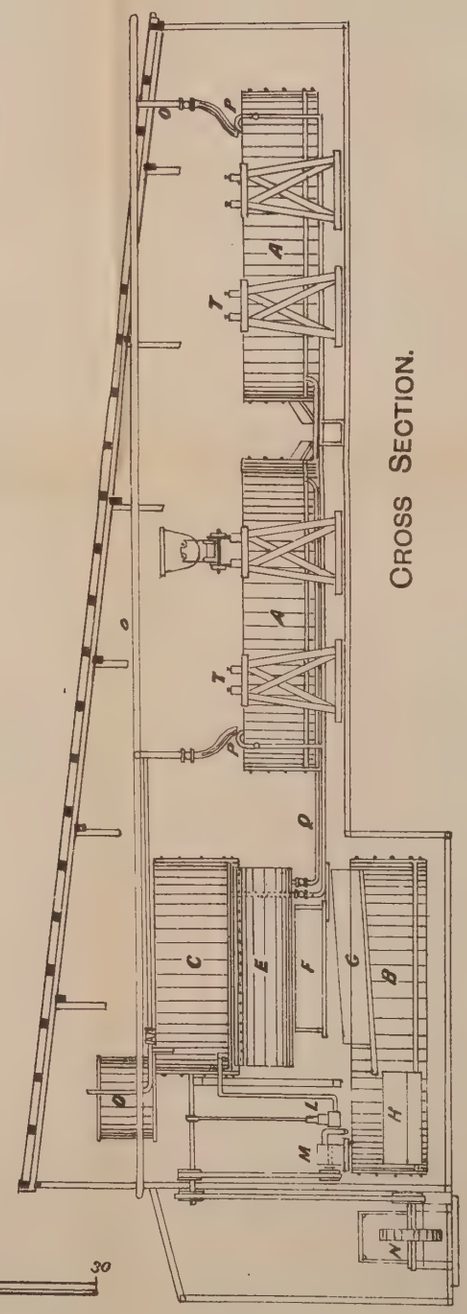
*A* Percolators
*B* Sumps
*C* Reservior
*D* Dissolving tank for Cyanide
*E* Vacuum chamber
*F* Tank to receive contents of *E*
*G* Zinc extractor boxes
*H* Slime settler
*I* Discharge launder
*J* Waste launder
*K* Strong liquor launder to extractor
*L* Air pumps
*M* Centrifugal pump
*N* Pelton wheel motor
*O* Fresh-water supply pipes
*P* Solution pipes—reservoir to percolators
*Q* Weak liquor pipes { percolators to vacuum chamber
*q* " " " { vacuum chamber to extractor
*R* Pump pipes
*S* Air-pump pipes
*T* Tramways over vats
*U* Melting room
*V* Assay room
*W* Laboratory
*X* Balance room
*Y* Dwelling room
*Z* Office




GROUND PLAN.

SCALE OF FEET.

10 5 0 10 20 30




CROSS SECTION.

to recover the coarser particles of gold which have resisted the dissolving action of the cyanide; and in both cases the tailings from the plates are concentrated for subsequent treatment.

The following interesting description of the Crown works was supplied to Mr. Henry A. Gordon, F.G.S., for his annual report for 1893, by Mr. J. McConnell, the general manager:—

**Battery Site.**—The battery site is situated on the side of the terrace on the south side of the Ohinemuri River, about 30 chains below the junction of the Waitawheta Creek. The foundations are cut out on the slope of the side of the hill, so that all the material is passed down stage after stage by gravitation as it is dealt with.

**Crushing Battery.**—The crushing battery consists of a Lamberton rock-breaker, and twenty heads of stamps of the American pattern, 900lb. each stamp. The building where the rock-breaker is placed is on trestle-work 45ft in height, strongly braced together. The ore as it is brought into the building is first dumped on to a grizzly, and what will not go through the bars of the grizzly runs down to the rock-breaker, and is broken up to a maximum size of 2in. in diameter, and then falls into the same hopper where the fine material goes that passes through the bars of the grizzly. It then passes from this hopper into the drying-kilns, which are built of brick, the hot air being confined in a long flue, having a series of steps to prevent the ore from travelling down too fast before it gets thoroughly dried. There is a cast-iron plate at the bottom of this flue which can be turned, to allow of the dried ore to pass down into a large hopper, made of steel-plates $\frac{5}{16}$in. thick, from which the Challenge ore-feeders are fed.

These kilns are only for drying the ore, and not in any way to calcine it. There are two of these kilns built on a stone foundation, and placed about 6ft. apart; the foundation going all the way across. The kilns themselves stand about 30ft. in height, the step flue being at an angle of about from 30° to 40° from the vertical. There is a furnace at the bottom, where either coal or firewood can be used to dry the ore.

**Stamp Mortars.**—There is first a concrete foundation put in for the stamps, and on the top of the concrete the stamp-mortars are each placed on the end of logs of kauri, each 18ft. in length, 4ft. 8in. one way, and 2ft. 2in. the other. These are firmly embedded in the concrete, and all bolted together so as to form a solid block of timber standing on end, having a length of 18ft. 8in. by a width of 2ft. 2in., and on this the four mortars are placed. They are fitted with screens, having the top standing outwards at a slight angle, and held to the face of the mortars by means of a long wedge, the gratings being 30-mesh, equal to 900 holes to the square inch.

**Stamps.**—The stamps are similar to those used by the Waihi Company, and are fitted with the latest appliances for raising and holding them up, the cams and tappets being all constructed on the American type. They are intended to make about ninety-two blows per minute, having a drop of 6in. The guides and framing are made of wood. Each ten-head battery is driven by a separate belt, and there is further provision made so that twenty additional stamps can be erected should they at any time be required. The pulverized material from the stamps falls into a chute and is conveyed into another set of hoppers at a lower level than the stamp mortars, and from these hoppers the pulverized dust is taken to the leaching vats.

**Cyanide Plant.**—This consists of twenty-four wooden vats, each 11ft. long by 9ft. wide and 3ft. 9in. deep. In the bottom of these vats there is a false bottom, or grating, placed about 3in. above the ordinary bottom, and on this false bottom a filter-bed is placed, about 4in. in thickness, the bottom layer being of coarse quartz-gravel, and gradually getting finer up to the top, the last coating being fine sand, having a coarse cloth placed over the top of the filter-bed to prevent the sand from being disturbed as the vats get cleaned out after every charge of pulverized ore. There are also fourteen agitators, eight of which are 5ft. deep by 4ft. 9in. in diameter, and six of them 6ft. deep and 5ft. 6in. in diameter. The agitators and vats are all made of kauri timber, the staves of the agitators being 3in. in thickness, and the vats being made of partly 3in. and partly 4in. timber, and all bolted together. Into each of these vats are placed three pipes, under the false bottom, so that the first, second, and third solutions can be drawn off into separate channels. On one side of each vat there is a door, which can be opened to admit of the material being sluiced out after the whole of the cyanide solution is completely washed out of the ore, the solution passing through a long series of boxes filled with zinc-shavings, which precipitates both the gold and silver in the form of a blackish powder. There are also three concrete sumps, each 15ft. by 12ft. and 6ft. deep, capable of holding about thirty tons of the cyanide solution; this is pumped up to the vats on the floor above as required. It is in these concrete sumps where the solution is always made up to the proper strength before being used.

**Agitation Plant.**—This plant consists of fourteen wooden tubs, fitted with revolving paddles, in which the ore and cyanide solution are agitated together until the

gold and silver is dissolved. The solution is then filtered, and the bullion deposited on the zinc-turnings, as already described.

Annexed are plans of the company's plant, to which the following description or reference applies:—At point *A* the ore is delivered at the battery, and tipped on to grizzly, *B*; the fines pass through and are conveyed to hopper, *D*; the roughs pass over the grizzly on to the stone-breaker floor, and are passed through stone-breaker, *C*, and fall into hopper underneath, marked *D*; the drying-kiln, *E*, is charged from this hopper. The ore, after passing through the kiln, being perfectly dry, is run into an iron hopper, *G*, from where it is automatically fed into stampers, *I*, by self-feeders, *H*; the ore, after passing through the stampers, is received in hoppers, *J*, and then conveyed by means of revolving tube, *K*, either into truck for conveying ore to agitation-cylinders for treatment, or, if the ore can be better treated by percolation, to store-hopper, *R*, in connection with percolation plant, from where it is trucked along the top of and tipped into percolation tanks, *S*, for treatment.

The plant is so arranged that the ore, after it is delivered above the stone-breaker, falls from the stage by gravitation, requiring the least possible handling, and thereby reducing the cost of labour to a minimum.

**Water Power.**—By means of a 4ft. by 2ft. fluming, the water is taken from a point in the Ohinemuri River above Karangahake, and conveyed to the battery, a distance of 85 chains, giving a fall there of 70ft., by which, with three Pelton wheels, 140-horse power can be obtained. The diameter of the water column is 44in.

Plate V.

# Plan of 40-Head Dry Stamper Battery,

ERECTED BY THE

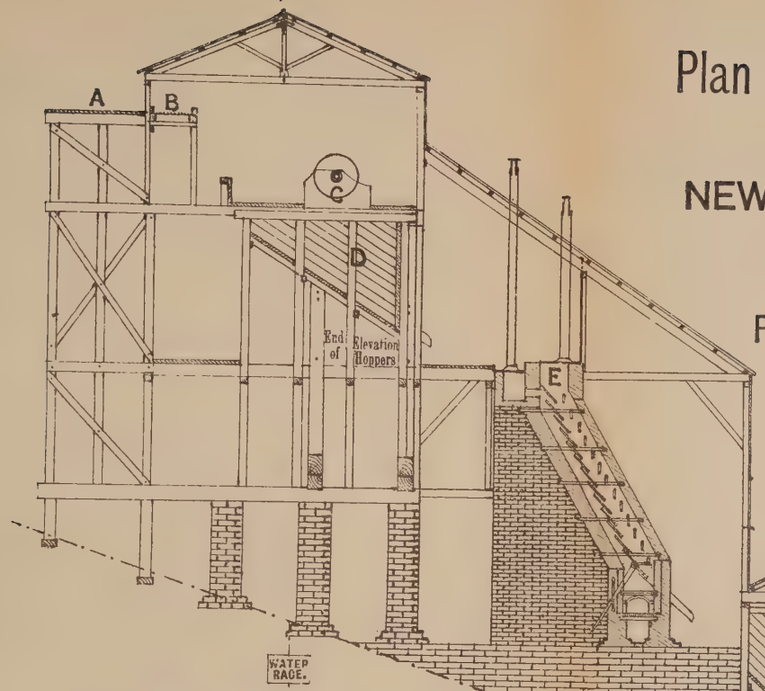
## NEW ZEALAND CROWN MINES CO.,

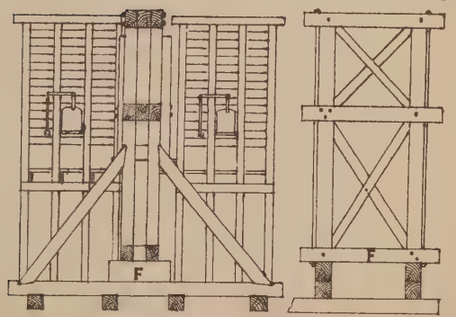
KARANGAHAKE.

For Treatment of Gold and Silver bearing Ores

by the

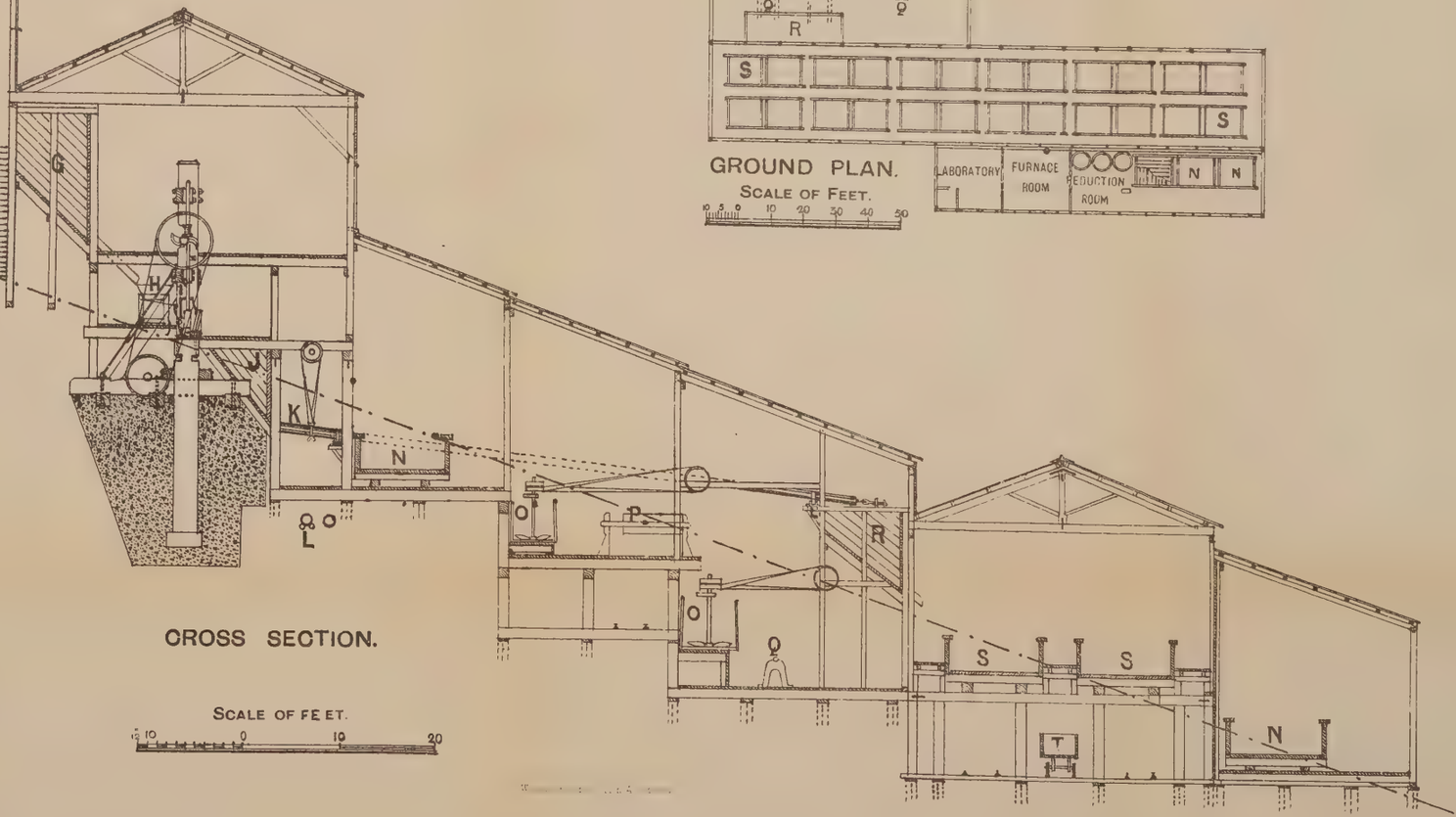
## McARTHUR-FORREST PROCESS.




FRONT ELEVATION OF HOPPERS.

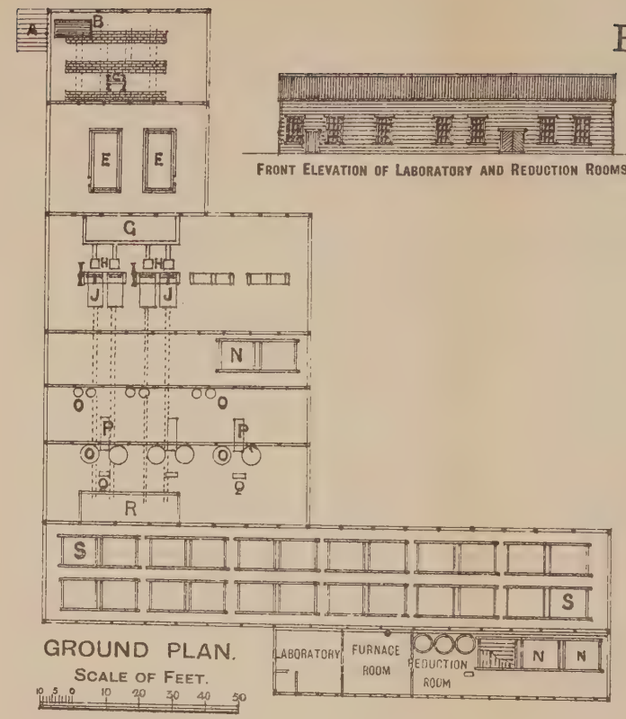
STONE BREAKER SECTION.




CROSS SECTION.

SCALE OF FEET.

10 0 10 20




GROUND PLAN.

SCALE OF FEET.

10 5 0 10 20 30 40 50



FRONT ELEVATION OF LABORATORY AND REDUCTION ROOMS.

## REFERENCE.

- **A** Tramway platform.
- **B** Ore grizzly.
- **C** Stone breaker.
- **D** Ore hopper.
- **E** Drying kiln.
- **F** Stone breaker frame.
- **G** Hopper for receiving ore for kiln.
- **H** Self feeder.
- **I** Stampers.
- **J** Receiving hopper for ore from battery.
- **K** Revolving tube } for conveying ore to agitation and percolation plant
- **L** Rollers for revolving tube.
- **M** End view of revolving tube.
- **N** Solution tanks.
- **O** Agitation vats.
- **P** Filter presses.
- **Q** Filter press pumps.
- **R** Store hopper for percolation lant.
- **S** Percolation tanks.
- **T** Truck for removing tailings

**Crushing Machinery.** – One Lamberton stone-breaker, capable of reducing 70 tons of ore per day fine enough to feed into stamps; and twenty heads of 9cwt. dry stamps, erected by A. and G. Price, which are guaranteed to crush 30 tons of ore per day sufficiently fine to pass through a 30-mesh screen.

The ore at present being treated is a hard, grey-coloured splintery quartz, quite free from all metallic impurities. A small proportion of the gold contents is too coarse for the action of cyanide, and is recovered on amalgamated copper plates, over which the tailings are slowly sluiced. The tailings beyond the plates are further concentrated by straight buddles.

The value of the ore varies from £5 to £7 per ton; and the rate of recovery is said to reach 90 per cent. of the original value, at a cost of 8s. per ton.

**Hutchison's Drying Furnace.**—The economical drying of the ore preparatory to dry-crushing is a matter of the highest importance, especially in districts where fuel is scarce. At most of the mills in the Hauraki goldfields the ore is dried, or calcined, in circular kilns excavated in the solid rock. At the New Zealand Crown works, at Karangahake, an improved step-furnace has been in use for some years, and has given great satisfaction.

The author is indebted to the courtesy of Mr. William Hutchison, metallurgical engineer to the Company, for the drawings of this drying-kiln, shown on Plate VI., and the following description:—

Mr. Hutchison states that the design is not new, nor altogether his own, the idea having been borrowed from the drying-kiln erected by the New Era syndicate at Waiorongomai, but which was so faulty as to be practically useless.

In the original kiln, only the lower side of the ore was exposed to the heat; but in the improved design, by means of a revolving door, or plate, marked *D*, at the bottom of the furnace, the heat is applied to both the bottom and top, thereby thoroughly drying the ore in half the time, and a much less consumption of fuel. The covering plate, marked *C*, is covered with pumice-sand to act as a non-conductor.

The revolving door, *D*, is fixed so that it can easily be removed should it become buckled or burnt through. A rod is fixed to the plate, and attached to a wire rope passing over two pulleys. There is a weight at the end of the rope to take the strain off the two moveable shafts on which the door revolves, or moves, when opening and shutting. Each plate is provided with a clearing hole, *E*, in case the finer particles that have passed over the grizzly should cake on the plates.

The capacity of the furnace is four tons, and the consumption of fuel about one ton of firewood for every eight tons of ore dried.

The number of bricks required is 15,000, and the cost of erection £400, which includes the iron-work, amounting to £120.

At the works of the Try Fluke Gold Mining Company, at Kuaotunu, the ore is wet-crushed through a 40-mesh screen, and passed over amalgamated copper-plates. The tailings are run directly from the plates into settling pits. When the pits are full, the slimes are removed from the lower end, and spread out to dry in the sun. When dry, they are broken up, and then filled into the leaching vats together with sand, in the proportion of one truck of slimes to two trucks of sand.

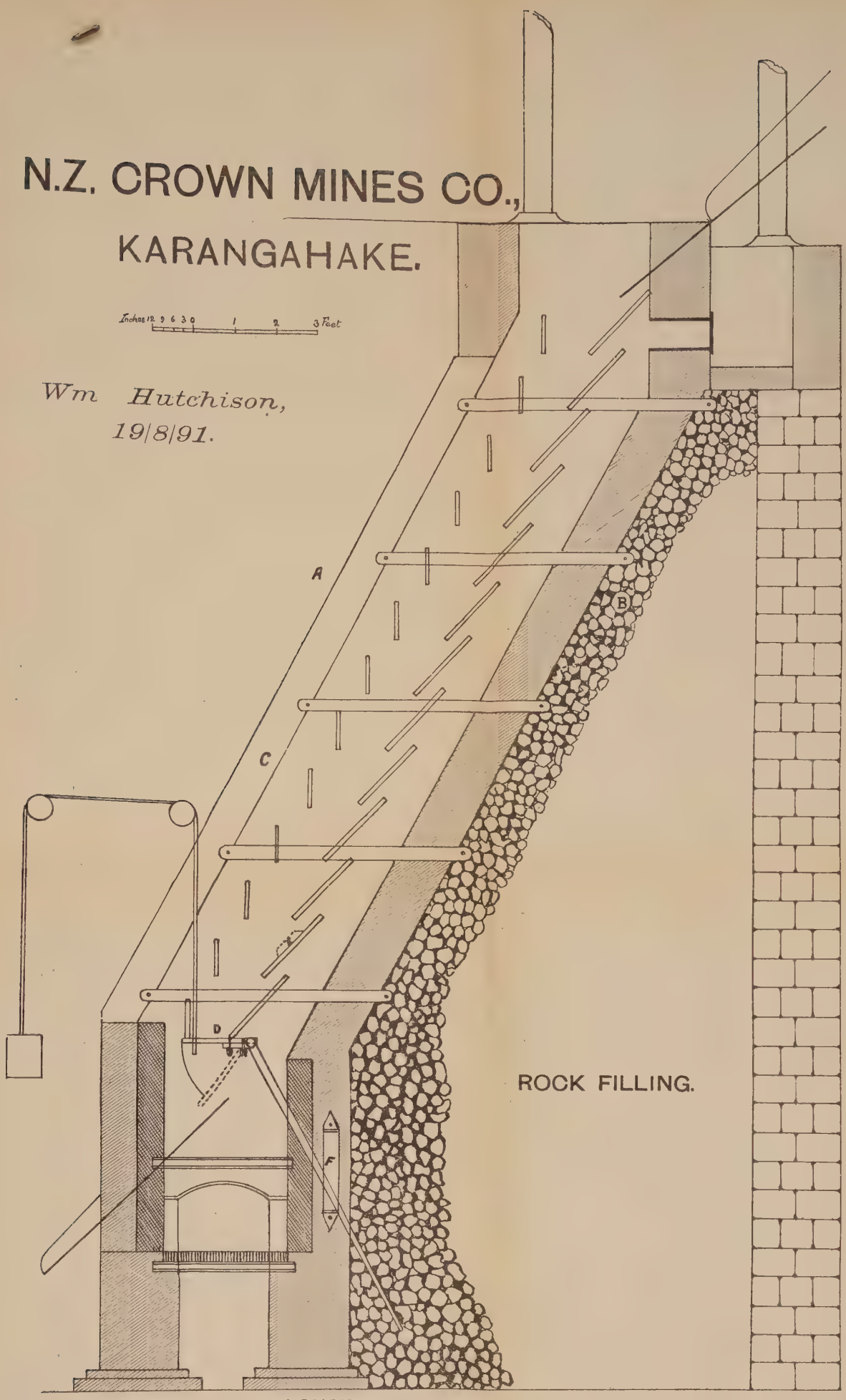
Plate VI.

# N.Z. CROWN MINES CO.,

## KARANGAHAKE.

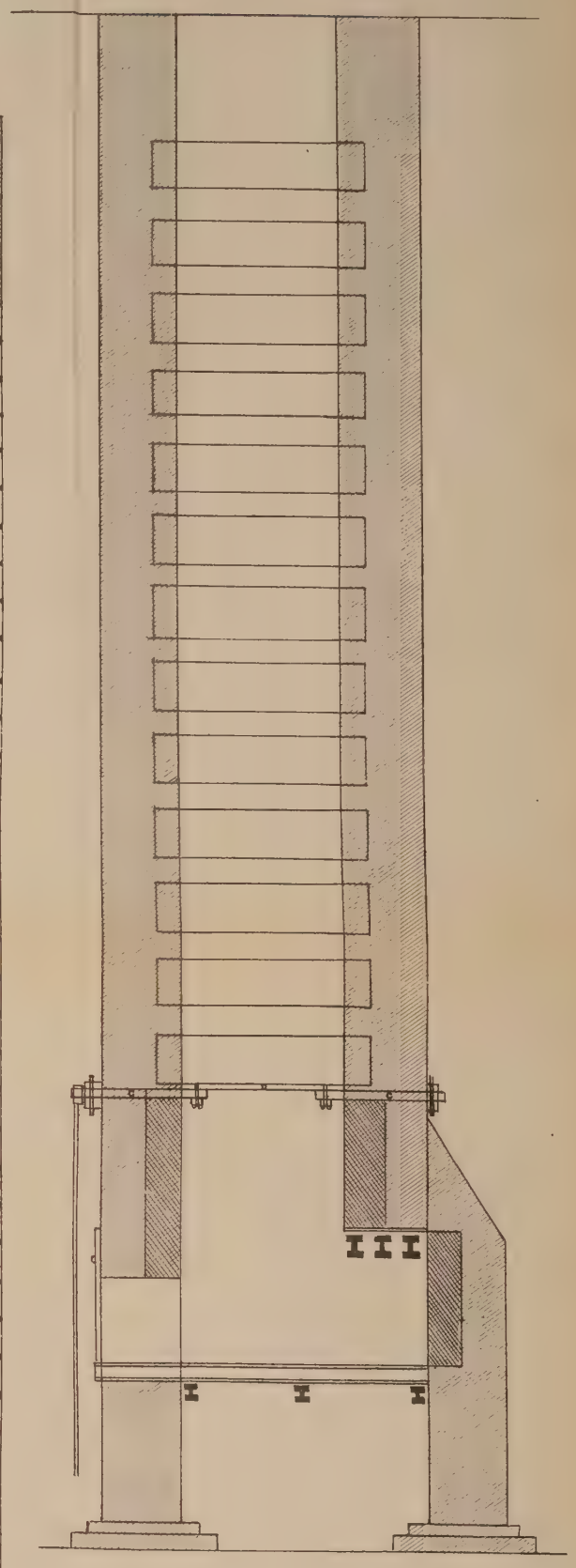
Inches 12 9 6 3 0 1 2 3 Feet

*Wm Hutchison,*
*19/8/91.*



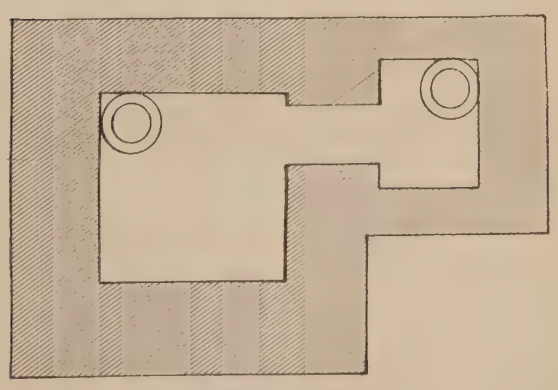

ROCK FILLING.

LONGITUDINAL SECTION.

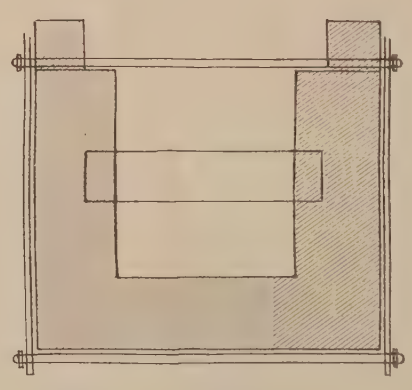


CROSS SECTION.

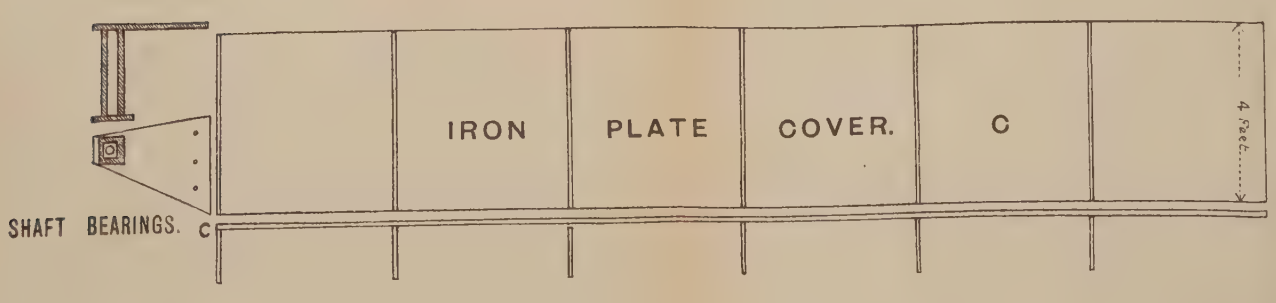
# DETAILS OF DRY KILN

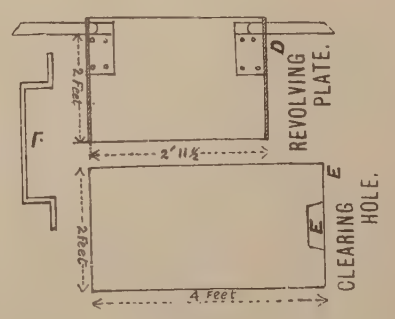


PLAN OF TOP.



SECTION THROUGH A.B.



The sand and slimes are thoroughly mixed in the vat before the solution is put on. Mr. F. E. Margetts, the Company's metallurgist, informs the author that the average time of treatment is as follows: —

| | HOURS. |
|---|---|
| Filling vats, 20 tons.. .. .. .. | 6 to 8 |
| Strong solution, 5 tons, 0·6 % KCy, standing in contact with tailings .. .. .. | 8 to 12 |
| Percolating .. .. .. .. | 24 to 30 |
| Weak solution, 5 tons, 0·2 % KCy .. .. | 5 to 6 |
| Weak cyanide washes, five of 5 tons each, 0·1 % of KCy .. .. .. .. | 25 to 30 |
| Totals .. .. .. .. | 68 to 86 |

The ore consists of grey and yellowish-brown quartz, sometimes containing a considerable proportion of iron and manganese oxides, the latter generally predominating. The greater part of the gold is excessively fine, being locally known as "float gold."

The average value of the tailings is about 20s. per ton; and the actual extraction by cyanide amounts to about 75 per cent., at a cost of 7s. 6d. per ton. If the tailings are of better value than usual, they are turned over in the vat after the last washing and again washed. In this case it is found that the extra extraction more than pays for the extra labour.

At the Kapai-Vermont mine, which adjoins the Try Fluke, the same ore is dry-crushed in a ball-mill, and then subjected to direct cyanide treatment with the most satisfactory results, the actual extraction generally exceeding 85 per cent. of the assay value. In this mine shoots of very rich ore have been frequently met with, containing a considerable proportion of comparatively coarse gold.

With such ores, the strong cyanide solution is circulated through the leaching vats until an adequate extraction is obtained.

**South Africa.**—At the Witwatersrand Goldfields the cyanide process is conducted on a more extensive scale than in New Zealand or elsewhere. The ore there is principally a pyritic, silicious quartz-conglomerate, consisting of rounded or sub-angular pebbles of bluish-grey quartz embedded in a quartzose matrix. The pyrites occurs in varying proportions in different mines, but the average is probably not less than 2 per cent. The gold does not exist in the quartz pebbles, but occurs disseminated throughout the matrix.

This ore is locally known as "banket," or almond rock. It is comparatively hard and somewhat splintery, and often contains a small proportion of corundum and clay, which render it tough and hard, and form slimy products during the crushing.

Throughout these goldfields the universal practice at present is wet-crushing with Californian stamps, copper-plate amalgamation, concentration or classification of tailings, cyanide treatment of tailings, and chlorination or cyanide treatment of concentrates.

At most of the batteries a 30-mesh screen is used, but in a few cases a finer or coarser mesh is adopted.

The main features of the cyanide treatment at the Witwatersrand Goldfields are much the same as those practised in New Zealand and Australia. The general working details are given below in tabulated form. They are the same at all the cyanide plants, with minor differences according to the individual fancy of the chemist or metallurgist in charge of the operations.

| | HOURS. |
|---|---|
| Filling vats .. .. .. .. .. | 12 |
| Preliminary alkali or water wash, if necessary .. | 12 |
| Strong solution, 0.4 °/₀ to 0.6 °/₀ KCy, ¼ to ⅓ of ore— | |
| In contact with tailings .. .. | 12 |
| Percolating .. .. .. .. | 12 |
| Dry or air-leaching .. .. .. .. | 4 |
| Weak solution, 0.15 °/₀ to 0.3 °/₀ KCy, about ¼ of ore | 12 |
| Dry or air-leaching .. .. .. .. | 4 |
| Two weak cyanide washes, 0.05 °/₀ to 0.1 °/₀ KCy, each about ⅛ of ore .. .. .. | 12 |
| Two clean water washes, each about ⅛ of ore .. | 12 |
| Discharging vat .. .. .. .. | 8 |
| Total .. .. .. .. | 100 |

The total quantity of solution used, including the water-washes, is about equal to the weight of the ore. The quantity of strong solution used varies according as a preliminary washing with a dilute solution has been employed or not. In the former case it is about 25 per cent. of the weight of the ore, and in the latter case about 40 per cent.

The percolation vats are charged with tailings to within a few inches of the top, and their surface is levelled. The strong cyanide solution is then allowed to penetrate the tailings until they are covered. The contents of the vat settle some inches, the amount of shrinkage depending on the depth of the vat and the percentage of moisture in the tailings.

The value of the tailings varies from 12s. to 20s. per ton, and the actual extraction amounts to 70 or 75 per cent., at a cost varying from 4s. to 10s. per ton, according to the size of the plant.

**Australia.**—During the past two years the cyanide process has been introduced, with marked success, in most of the Australian colonies.

In Queensland, at Charters Towers, there is a plant capable of treating 800 tons per month. It is a Customs plant, the principal material treated being sludges purchased from the surrounding mines. These sludges are the concentrated residues resulting from grinding and amalgamation in berdan pans.

At Croydon, in the same colony, there are several tailings plants in operation. The results have been most successful, the gold produced having largely increased the output from that goldfield.

In New South Wales, at Mitchell's Creek, there is a tailings plant capable of treating 2,000 tons per month, erected under the auspices of the Australian Gold Recovery Company, Limited. Several other cyanide plants are now in course of erection, principally for the treatment of tailings.

In Victoria, a plant to treat 2,000 tons per month has been constructed at the New Golden Mountain Gold Mining Company's property, for the direct treatment of the ore by cyanide. No particulars of the operations at this plant are available.

South Australia has several cyanide plants in operation. The plant at Mount Torrens is a Government Customs plant, erected for experimental purposes. Much good work has already been done there. At the Virginia Company's cyanide works, the sumps are built of cement, their dimensions being 16ft. by 14ft. by 12ft.

**America.**—The cyanide process has been introduced into the States of Utah, Montana, Colorado, South Dakota,

California, and also Mexico. The results in some cases have been most successful, but generally speaking the adoption of the cyanide process in the States has been very slow. The primary reason for this is no doubt due to the circumstance that the ores are not so well adapted for cyanide treatment as those of South Africa and New Zealand, where the process has made greater strides than elsewhere.

One of the most successful plants is that of the Mercur Gold Mining and Milling Company, in Utah. The ore is a silicious surface-ore, with the gold very finely divided. It passes through a Dodge rock-breaker, and is then coarsely crushed by two sets of Wall's corrugated rolls; the first set to one half-inch; the second to one-quarter inch.

This coarsely-crushed ore is leached by percolation in vats holding 14 tons. The cyanide solution is generally allowed to stand in contact with the charge for 12 hours; and then allowed to percolate from 36 to 48 hours. The extraction amounts to over 85 per cent., at a cost of 9s. per ton.

Another very successful plant is that at Revenue, Madison County, Montana. The Revenue ore is nearly pure silica, containing from 1 to 2 per cent. of iron peroxide, and no sulphides. The best amalgamation methods only saved from 25 to 27 per cent. of the assay value, while the cyanide process recovers from 80 to 87 per cent.

The ore is crushed and passed through a 30-mesh screen. The leaching vats are 10ft. and 12ft. in diameter, and 4ft. and 4½ft. deep, of 3in. pine. The strength of the solution varies from 0·6 per cent. to 1·0 per cent. The

percolation takes from 24 to 36 hours, after which the solution, now containing from 0·4 per cent. to 0·8 per cent. of cyanide, is pumped back for washing. Half-a-ton of strong solution is used for every ton of ore. The consumption of zinc is about half-a-pound per ton of ore.

Sulphuric acid is used for refining the precipitates. The total cost of treatment is about 16s. per ton, not including patent-royalty.

## CHAPTER VIII.

### LEACHING BY AGITATION.

The first attempt to introduce the cyanide process on a working scale, for the recovery of gold and silver from their ores, was made by the Cassel Gold Extracting Company, of Glasgow, at the New Zealand Crown mines, Karangahake, in 1889. The operations were under the supervision of Mr. J. MacConnell. In the first plant, agitation formed a prominent feature, but in later years leaching by percolation became the more general and favourite method of treatment.

The new and extensive cyanide works of the Crown Mines Company, besides a percolation plant of twenty-four tanks, contain an agitation plant consisting of sixteen wooden tubs, or barrels, fitted with revolving paddles. The agitators are seldom used, preference being given to percolation.

At the late concentration plant at the Sylvia mine, Thames Goldfield, an agitation plant was erected by Dr. A. Scheidel, in 1891, consisting of three agitators, 6ft. in diameter and 6ft. deep; three vacuum filters, together with the necessary solution tanks; zinc extractors, and other appliances for cyanide treatment.

The ore was heavily charged with iron pyrites, and occasionally a small proportion of zinc-blende, argentiferous galena, and copper pyrites. It was wet-crushed in a 10-stamp Californian battery, classified in pyramidal boxes, and subsequently concentrated in jiggers, slime-tables, and buddles.

The concentrates of four grades were afterwards subjected to cyanide treatment by agitation. The extraction from the best slimes is said to have amounted to 96·45 per

cent. of the gold, and 94·59 per cent. of the silver. The average extraction from all classes of concentrates amounted to 82·67 per cent. of the assay value.*

The strength of the cyanide solutions varied from 0·5 per cent. to 1 per cent., and the time of agitation from six to twenty-four hours. About 300 tons of concentrates yielded over £10,000 value of bullion, but the cost per ton is not given.

The extraction is said to have been satisfactory until an excess of copper ore appeared in the concentrates, which rendered them unsuitable for cyanide treatment.

The cyanide plant at the Thames School of Mines, designed by the author, is provided with an agitator which serves a double purpose, being used as an agitator and also as a dissolving vat when ores are being treated by per colation.

The agitator is similar to those used at the Crown and Sylvia works, consisting of an upright tub or barrel provided with a central revolving spindle set in a foot-step at the bottom. At the bottom end of the spindle is fixed a screw, consisting of four paddles or blades. The foot-step being in the agitator is subject to great wear and tear, and this forms a most objectionable feature, as it must be continually renewed. This difficulty is easily got over by fixing a hollow cone in the centre of the agitator and placing the spindle in this, the motion being applied from below, as in most grinding and amalgamating pans. An agitator of this kind was erected in 1894 by Dr. Scheidel at the Utica mine, Calaveras County, in California.†

* The Cyanide Process, by Dr. A. Scheidel, p. 79.

† *Ibid*, 1894, p. 89.

**The Actual Extraction by Agitation.** — The practice of the author is to charge the agitator with the cyanide solution made up to the required strength, using from 40 per cent. to 60 per cent. of the weight of the ore. The agitator is then set going, about 50 revolutions per minute, and the ore or tailings then gradually fed in, until the charge is complete. The agitation is continued until a satisfactory extraction has been effected, which generally takes from six to ten hours. Samples for assay can be obtained from the agitator by means of a small tin at the end of a stout string. When the extraction is considered adequate, the agitator being still in motion, the stop-cock is opened and the pulp allowed to discharge into a percolation vat where the solution is drawn off and the residues washed, aided by an artificial vacuum. The cyanide solutions are then passed through the zinc precipitation boxes.

Leaching by agitation possesses many advantages over percolation for the treatment of pyritic concentrates or rich tailings. The dissolution of the gold is much more expeditious, taking hours where percolation requires days; and with suitable material the extraction is always high.

On the other hand, agitation requires motive-power, and from the nature of the process the charges must be small, in no case exceeding a few tons. There is a prevalent belief that agitation causes an excessive consumption of cyanide by decomposition by atmospheric carbonic acid gas; but the author thinks this source of loss is much exaggerated, and is certainly much less than it was in the early attempts at cyanide treatment when agitation was prolonged from 36 to 48 hours continuously.

The author has found, by many trials, that from six to eight hours' agitation is sufficient to effect the dissolution of

the gold in the most refractory ores when reduced to a sufficient degree of fineness.

Numerous experiments during the progress of the agitation have shown that the greater portion of the gold was extracted during the first hour.

The rate of the extraction at the different hour-periods, during the treatment of the Monowai sulphide ore, are given below, and will be found instructive :—

| TIME OF AGITATION. | GOLD EXTRACTION °/₀ |
|---|---|
| After 1 hour | 85.4 |
| ,, 2 ,, | 88.2 |
| ,, 3 ,, | 90 6 |
| ,, 4 ,, | 92.3 |
| ,, 5 ,, | 92.3 |
| ,, 6 ,, | 92.3 |

From the above it will be seen that the maximum extraction was obtained in four hours. This was a complex sulphide ore, containing sulphides of copper, zinc, iron, and lead. An analysis of the more mineralized portion by the author's assistant, Mr. F. B. Allen, M.A., B. Sc., gave the following results :—

| | |
|---|---|
| Insoluble gangue .. .. .. .. | 90.15 |
| Copper pyrites .. .. .. .. | 3.78 |
| Iron pyrites .. .. .. .. | 4.40 |
| Galena .. .. .. .. .. | 0.25 |
| Zinc-blende .. .. .. .. | 0.26 |
| Alumina .. .. .. .. | 0.13 |
| Water and loss .. .. .. .. | 1.03 |
| | 100.00 |

The bullion contents of this ore were gold, 1oz. 5dwts., and silver, 14oz., per ton. A 0·6 per cent. solution of cyanide was used for leaching by agitation, and the consumption amounted to 0.45 per cent., with an extraction of 92 per cent. A very large amount of copper was dissolved, and, becoming deposited on the zinc, caused much trouble in the precipitation of the bullion. The actual extraction was below 70 per cent.

The following interesting and instructive experiments by agitation with 0·25 per cent. and 0·33 per cent. cyanide solutions on "Martha" ore were kindly supplied by Mr. E. G. Banks, the metallurgist for the Waihi Gold Mining Company at Waihi.

EXPERIMENTS *on "Martha" ore, showing rate at which the precious metals were extracted by a* 0·25 *per cent. solution of KCy, and amount of KCy consumed.*

| | | GOLD. oz. dwt. gr. | SILVER. oz. dwt. gr. | SAVED % GOLD. | SAVED % SILVER. | KCy USED. |
|---|---|---|---|---|---|---|
| ORE BEFORE TREATMENT | | 0 16 8 | 3 4 13 | | | |
| 1. ,, | after 2 hrs. agitation | 0 9 19 | 2 15 20 | 40 1 | 13 5 | ·08% |
| 2. ,, | ,, 4 ,, | 0 8 4 | 2 13 22 | 50·0 | 16·5 | ·09% |
| 3. ,, | ,, 6 ,, | 0 6 14 | 2 7 10 | 59·7 | 26·5 | ·094% |
| 4. ,, | ,, 8 ,, | 0 4 14 | 2 4 10 | 71·4 | 31 2 | ·103% |
| 5. ,, | ,, 10 ,, | 0 3 12 | 1 19 20 | 78·6 | 38 3 | 117% |
| 6. ,, | ,, 12 ,, | 0 3 12 | 2 3 1 | 78·6 | 33 9 | ·132% |
| 7. ,, | ,, 14 ,, | 0 3 6 | 2 2 11 | 80·1 | 34 2 | ·126% |
| 8. ,, | ,, 16 ,, | 0 3 6 | 2 2 11 | 80·1 | 34 2 | ·13% |
| 9. ,, | ,, 18 ,, | 0 3 10 | 2 3 3 | 79·1 | 33·2 | ·13% |
| 10. ,, | ,, 20 ,, | 0 3 1 | 2 4 8 | 81·4 | 31·3 | ·13% |
| 11. ,, | ,, 22 ,, | 0 2 15 | 2 2 8 | 83·9 | 34·4 | ·135% |
| 12. ,, | ,, 24 ,, | 0 2 11 | 2 2 11 | 84·3 | 34·2 | ·135% |

FINENESS OF ORE TREATED ... 1·5 % remained on a 40-mesh screen
14·5 % ,, 60 ,,
26 % ,, 80 ,,

SIMILAR EXPERIMENTS *with a 0·33 per cent. solution of KCy.*

| | | GOLD. oz. dwt. gr. | SILVER. oz. dwt. gr. | SAVED % Au. | SAVED % Ag. | KCy USED. |
|---|---|---|---|---|---|---|
| ORE BEFORE TREATMENT | | 0 17 4 | 3 11 22 | | | |
| 1. ,, after 2 hrs. | agitation | 0 9 2 | 2 16 2 | 47·1 | 22 0 | ·09% |
| 2. ,, ,, 4 | ,, | 0 7 21 | 2 13 19 | 54·1 | 25·2 | ·093% |
| 3. ,, ,, 6 | ,, | 0 5 20 | 2 9 18 | 66·0 | 30·8 | ·093% |
| 4. ,, ,, 8 | ,, | 0 4 9 | 2 5 18 | 74 5 | 36·4 | ·106% |
| 5. ,, ,, 10 | ,, | 0 3 5 | 2 2 21 | 81·3 | 40·4 | ·119% |
| 6. ,, ,, 12 | ,, | 0 2 2 | 2 1 8 | 87·8 | 42·5 | ·126% |
| 7. ,, ,, 14 | ,, | 0 2 2 | 1 19 10 | 87 8 | 45 2 | ·138% |
| 8. ,, ,, 16 | ,, | 0 1 20 | 1 16 18 | 89 3 | 48·9 | ·14% |
| 9. ,, ,, 18 | ,, | 0 1 14 | 1 17 16 | 90·8 | 47·6 | ·14% |
| 10. ,, ,, 20 | ,, | 0 1 9 | 1 16 1 | 91 9 | 49 9 | .154% |
| 11. ,, ,, 22 | ,, | 0 1 4 | 1 14 19 | 93 2 | 51 6 | ·154% |
| 12. ,, ,, 24 | ,, | 0 1 4 | 1 14 0 | 93 2 | 52·7 | ·16% |

FINENESS OF ORE TREATED ... 2·1 % remained on a 40-mesh screen
17 % ,, 60 ,,
29·7 % ,, 80 ,,

The results of Mr. Banks' experiments show that in the case of the 0·25 per cent. solution the commercial extraction was obtained in 20 hours, and with the 0·33 per cent. solution in 12 hours. The consumption of cyanide in each case was about the same, while as high an extraction was obtained with the stronger solution in 10 hours as with the weaker solution in 20 hours. The results clearly show that the maximum rate of dissolution was reached with the 0 33 per cent. solution.

## CHAPTER IX.

### ZINC PRECIPITATION OF GOLD.

The zinc for bullion precipitation is used in thread-like turnings, as this form gives the most surface for the least weight. It should be the best obtainable, and free from arsenic or antimony, although a little lead is an advantage, as it causes more rapid precipitation by forming a voltaic pair with the zinc.

As a general rule, one cubic foot of zinc turnings will precipitate the gold from two tons of solution, in twenty-four hours.

Zinc on which a film of bullion has been precipitated is more active than pure zinc, and it is therefore advisable to replace the zinc dissolved in the upper compartments by moving the zinc forward from the lower compartments, and adding fresh zinc to the latter.

In practice, the cyanide solution is allowed to slowly drain through the zinc in the precipitating boxes. The rate of flow is soon determined by actual experience. It is generally found that 85 per cent. to 95 per cent. of the bullion will be precipitated in the first three boxes.

The solution, after leaving the boxes, should not contain more than six or eight grains of gold to the ton.

The principle of the precipitation of gold by metallic zinc is based on the fact that cyanide has a stronger affinity for zinc than for gold, as shown by the following equation :—

$$2Au\,KCy_2 + Zn = Zn\,K_2\,Cy_4 + 2Au.$$

By the above reaction it will be seen that 1oz. of zinc should precipitate 6oz. of gold, but in practice it is found that from 4oz. to 12oz. of zinc are required to precipitate 1oz. of gold. The reactions which take place in the zinc precipitating boxes are at times most varied and perplexing, especially during the treatment of pyritic tailings or acid mineralized ores.

Part of the excessive consumption of zinc is no doubt due to decomposition by free cyanide, as may be ascertained by testing the solution for available cyanide before entering and after leaving the zinc precipitating boxes; but the consumption and consequent loss of cyanide by this cause is much less than generally supposed, and in all cases insufficient to account for the great waste of zinc.

The precipitation of the gold, doubtless from electro-chemical causes, is always more rapid and complete from moderately strong than from very weak cyanide solutions.

It has been suggested by some chemists that this is due to nascent hydrogen, liberated by the action of the free KCy on the zinc, taking the place of the gold according to the following equations:—

$$4KCy + Zn + 2H_2O = ZnK_2Cy_4 + 2KHO + H_2.$$

and

$$2AuKCy_2 + H_2 = 2KCy + 2HCy + Au_2.$$

The liberated hydrocyanic acid is capable of combining with any free alkali present, and thus there would be no loss of the cyanogen combined with the gold. This reaction is shown by the following equation:—

$$HCy + NaHO = NaCy + H_2O.$$

Hydrogen gas is always evolved when gold is precipitated, and the gentle action of the gas bubbles, as they

rise to the surface in the zinc boxes, is always an indication of satisfactory precipitation.

During the treatment of pyritic tailings at Kuaotunu, the unsatisfactory precipitation of the gold was for some time a source of much trouble to the chemists in charge of the cyanide works, but this difficulty was overcome by making up the strength of the solution before entering the extractor to something like the original working strength. This course was adopted at the suggestion of Mr. J. A. Walker.

In practice, it is effected without any extra trouble by simply placing a barrel containing a strong solution of cyanide at the head of the extractor, and allowing a steady drip into the cyanide solution, in the top compartment, which is filled with a filter of sand and gravel. By testing the cyanide solution a few times, the rate of drip to bring it up to the required strength can easily be determined.

The author used this method with great success in the treatment of the cupriferous ores from the Monowai mine in the Hauraki Goldfields.

It was found that the dissolved copper was precipitated much more rapidly from a weak solution of cyanide than from a strong one. In order to overcome this the solutions were made up to the original working strength.

When copper is present in the solution, it soon covers the zinc with a bright metallic coating, which begins in the lower boxes, and gradually encroaches on the upper ones. When the zinc is coated with copper, the precipitation of the gold is very slow and imperfect. By increasing the strength of the solution to near the working strength, before it enters the boxes, the copper may be largely kept in solution.

If the zinc turnings be placed in a solution of lead acetate, of say 1 per cent. strength, they will become covered with a porous coating of lead. This lead-coated zinc, by its electro-chemical energy, will effect the perfect precipitation of the gold, and leave the copper in solution. The resulting bullion obtained by this means, is, however, always highly charged with lead.

Experience has shown that ores containing copper are not adapted for cyanide treatment, firstly, on account of the undue consumption of the cyanide; and secondly, on account of the difficulty of precipitating the gold in the presence of the base metal; moreover, by continued use, the stock and sump solutions become charged with copper, and thus rendered useless for all practical purposes, such as washing, or forming the basis of working solutions.

Occasionally a gritty, greyish-white, porous precipitate of zinc cyanide forms on the zinc in the precipitating boxes. The reactions which lead to its formation have not yet been satisfactorily explained, but, whatever they may be, its presence is always accompanied by imperfect precipitation of the gold. This precipitate of zinc cyanide is seldom seen excepting in the treatment of pyritic ores and tailings. It can generally be prevented by a careful preliminary washing, and treatment with lime instead of caustic soda.

In some cases there may be, in the presence of organic compounds, an excessive and injurious evolution of hydrogen. During the treatment of decomposing pyritic tailings at the Great Mercury Cyanide Works, Kuaotunu, the evolution of hydrogen gas was so vigorous that it lifted the zinc out of the precipitation boxes, forming a thick froth.

On this occasion the precipitation of the bullion was very imperfect and unsatisfactory, and was suggestive of polarization.

The precipitation of gold by zinc, results in the formation of a double cyanide of zinc and potassium, and the continual use of the same solutions would lead to the belief that the working solutions would in time become charged with zinc salts. In actual practice it is found that this is not the case to any great extent.

Feldtmann states, that under favourable conditions, the zinc-potassic cyanide is of itself capable of dissolving gold from its ores. He considers that the small quantities of alkaline sulphides present in commercial cyanide, or formed by the action of cyanide on metallic sulphides, serve to precipitate a portion of the zinc as the insoluble sulphide.

In cases where ores contain considerable proportions of metallic sulphides, soluble in cyanide solutions, sufficient alkaline sulphide may be formed to precipitate a portion of the dissolved gold.

To prevent any loss in this direction, Mr. J. S. MacArthur suggests the addition of a solution of some soluble lead or other metallic salt which is known to form an insoluble sulphide in alkaline solutions. In this case, however, it would be advisable to avoid an excess of the lead, or other salt, so as to prevent possible complications in the extractor. The exact amount of salt required could be determined in the laboratory.

**The Clean-up.**—The periodical clean-up takes place once or twice a month. The first operation is to pass a current of clean water through the zinc boxes, so as to remove the cyanide solution, which is injurious to the workmen, often causing their arms to become covered with painful red boils.

The trays holding the zinc are then moved up and down in their compartments so as to allow the fine gold

precipitates, and fine particles of zinc, to fall through the sieve and settle in the bottom of the box. The contents of the trays are then placed in a large trough, provided with an easily removable false-bottom of finely perforated iron. The zinc is gently teased out and rubbed in this trough, which is partly filled with clean water, and in this manner as much as possible of the adhering gold is removed. After all the gold has settled, as a slimy mass, the water is syphoned off. The gold slimes remaining in the extractor are sluiced through plug-holes into the side launder and collected in a trough. The fine slimes or precipitates are rapidly settled by the addition of a little powdered alum to the solution.

In large cyanide works the precipitates are dried in a small vacuum-filter. The discoloured zinc shavings are now returned to the precipitation boxes, fresh zinc being put in the lower compartments. The gold still remaining on the zinc is recovered at the next clean up.

**Roasting the Precipitates.**—The dry precipitates are roasted at a low heat, with free access of air. The object of the roasting is to oxidize the zinc in the slimes, and thus cause it to combine with the fluxes in the subsequent smelting, and thereby leave the bullion as fine as possible.

In New Zealand cyanide works, the roasting furnace consists of a large flat cast-iron plate, with raised edges. It is built over a small grate or furnace, and a hood of light sheet-iron is placed over the roasting-plate, so as to carry off the zinc fumes.

The roasting should be conducted at a moderate heat, *i.e.*, it must never rise above a *dull red*, and the precipitates must be stirred continuously so as to expose fresh surfaces to the action of the atmospheric oxygen. During the early

part of the roasting, dense white vapours of zinc oxide are given off, but as the operation advances they are observed to diminish and finally to cease entirely when the reaction is complete. Time, from one to two hours.

Mr. Feldtmann has found that the oxidation of the zinc is facilitated by the addition of a little nitre, say from 3 per cent. to 10 per cent. He suggests that it should be applied to the slimes as a strong solution before the drying, so as to get thoroughly mixed with the whole mass. The nitre not only helps to oxidize the zinc, but is also said to assist the subsequent fluxing by unity with the zinc oxide, and forming a zincate of potash, which is not so readily reduced by the plumbago of the crucible as the oxide. At many works the nitre is added to the dried slimes in a powdered form; of course less nitre must be used than is necessary to oxidize all the base metal present, as any free nitre remaining would rapidly destroy the plumbago crucible during the smelting process. Besides rendering the bullion fine, the nitre roasting gives a cleaner slag and greatly expedites the fusion.

When stirring and removing the roasted precipitates, care must be exercised, to avoid a loss of bullion in the form of dust.

**Smelting the Oxidized Precipitates.**—The roasted precipitates are now placed on a large, shallow iron tray, mixed with the necessary fluxes, and fused in plumbago crucibles. The following fusing mixtures have always given satisfactory results:—

| | Clean Precipitates Little Zinc. | Much Zinc and Little Sand. | Very Sandy. |
|---|---|---|---|
| Precipitates | 100 | 100 | 100 |
| Bicarbonate of Soda | 6 | 20 | 50 |
| Borax | 50 | 50 | 30 |
| Sand | 3 | 15 | — |
| Fluor-spar | — | — | 2 |

The sand is added to form a fusible slag with the soda, and to protect the crucibles from metallic oxides and the potash formed by the reduction of the nitre.

In works where large quantities of precipitates have to be smelted, Nos. 50, 60, or 70 plumbago crucibles will be required.

**The Actual Fusion.**—The crucible, previously annealed, is placed on a flat brick resting on the bars of the furnace. A priming of borax is then placed in the crucible, and over this a charge of precipitates; fresh additions of precipitates are made as the charge fuses and subsides. When the crucible is two-thirds full, the slag is skimmed off and fresh portions of precipitates added until it is three-fourths full of molten bullion.

The crucible is now removed from the furnace, and the contents poured into ingot moulds which have been previously *well heated* and carefully oiled with the best olive oil. All excess of oil should be wiped out of the mould before pouring the metal.

The melting furnace may be constructed to hold two or three crucibles at the same time. It should be built of the best materials, as the heat required to melt the *slime mixture* is higher than that for ordinary melting.

At the Langlaagte Cyanide Works at Johannesburg, the slimes, mixed with the fluxes, are charged into No. 50 plumbago crucibles, and melted in a reverberatory hearth furnace which holds 22 crucibles at the same time.

The time required for melting varies from one and a-half to three hours, according to the character of the materials and the temperature of the furnace.

The slag resulting from the smelting of slimes always contains a small proportion of gold. It is, therefore,

generally pulverized in a single stamper, or in a small mill, and afterwards amalgamated with mercury.

The ingots of bullion, obtained from the first smelting, are re-melted with borax; and, since gold forms but a very imperfect alloy with zinc, this second melting should be conducted at as low a temperature as possible so as to obtain an approximately uniform bar of bullion.

The zinc slimes generally contain from 30 per cent. to 65 per cent. of bullion, the fineness of which, after melting, generally varies from 600 to 900.

The clips for assaying should be taken from different parts of the bar so as to obtain a representative sample for valuation.

**Refining by Sulphuric Acid.**—This method is used a good deal in cyanide works in America, but has not yet been adopted in New Zealand. The acid treatment of the precipitates is a simple enough operation, and is occasionally used by the author for the refining of small parcels. The necessary apparatus consists of three shallow wooden tubs.

The operation is conducted as follows:—Clean water is passed through the zinc-extractor, for some time, to remove all trace of cyanide. The precipitates are then removed from the boxes and placed in the first tub, with a sufficient quantity of dilute sulphuric acid. The acid should not be too strong, nor yet too weak; a mixture consisting of five parts of water and one part of strong acid answers well.

The quantity of dilute acid will depend on the proportion of zinc present in the precipitates. With 50 per cent. of zinc, about six parts of the acid mixture to one of the precipitates will be required; and with very zincy precipitates, from ten to twelve parts.

The mass in the tub should be stirred occasionally, and then allowed to settle. Heat is generated, and large quantities of hydrogen gas liberated, by the action of the acid on the zinc.

When the undissolved precipitates have been allowed to settle, the clear liquid should be removed by decantation into the second tub, and thence finally, after an interval, into the third tub. By this means any fine particles of bullion which have escaped in the first decantation, will be secured in the second tub; and that which has escaped during the second decantation will be found as a fine sediment in the third tub.

The bullion should now be washed in the tubs with clean water, to remove all base soluble sulphates and any free acid remaining. Then remove the bullion slimes and dry on the vacuum-filter. When dry, subject to an oxidizing roasting on a shallow iron pan, for an hour or so, to oxidize any base sulphates present.

Next, mix with 5 or 10 per cent. of borax-glass, according to the amount of zinc oxide still present, and fuse in a plumbago crucible in which a priming of borax has already been placed. As the charge fuses and subsides, fresh portions of bullion should be added, until the crucible is three parts full of melted metal. To permit this the slag can be skimmed off from time to time.

The bullion is generally from 850 to 900 fine, but with a little extra trouble can be worked up to 950.

With suitable appliances, this process possesses many advantages over the smelting process. It occupies less time, produces finer bullion, and, properly conducted, costs less.

When large quantities of precipitates have to be dealt with, the method of settling the slimes and decanting is

too slow and expensive. In this case, the separation of the slimes from the acid solution, as well as the subsequent washings, must be effected in a vacuum-filter.

The filter used for the purpose is a wooden box, two or three feet square. It is provided with a filter webbing, or cloth, of fine canvas or twill duck, resting on a grating of wood, and fixed with slips of wood, so as to be easily detached for washing.

The false-bottom, below the webbing, must be 15in. or 20in. deep, and provided with a solution guage, the upper limb of which should be 2in. below the air-exhaust pipe connected with the vacuum boiler.

Care must be taken to draw off the acid solution and washings by a plug-hole before they rise to the level of the air-exhaust pipe, which is placed immediately before the filter-frame.

When the acid solution is diluted to half its strength before filtering, the webbing lasts for several operations.

## CHAPTER X.

### THE SIEMENS-HALSKE PROCESS.

The distinguishing features of this process are the use of extremely dilute solutions of cyanide and the electrical precipitation of the gold.

Since the introduction of the cyanide process, the precipitation of the gold by metallic zinc has always been regarded as a weak point; and metallurgists have devoted much time in the endeavour to discover an efficient substitute for it.

Electrical precipitation naturally engaged the attention of many investigators. In 1893, the author, assisted by Mr. F. B. Allen, M.A., B.Sc., of the Thames School of Mines, conducted a number of experiments with electrical precipitation to determine the method of precipitation to be adopted at the School of Mines Cyanide Plant. Many different modifications were tried. With some, the precipitation from solutions of ordinary working strength was very satisfactory; but, with all, the precipitation of the gold from dilute solutions, such as those corresponding to weak cyanide washes, was always very imperfect and accompanied with the decomposition of the water.

In the Siemens-Halske process this difficulty is overcome by causing a slow artificial circulation of the cyanide solutions in the extractor.

The plant and operations connected with the leaching of the gold are the same as those described in the preceding chapters, the only difference being in the extractor-house.

The electrical precipitation of gold has been introduced with marked success at a number of cyanide plants at the Witwatersrand Goldfields, and its use is extending. Up to the present it has not been introduced to New Zealand or Australia, and so far very little has been written about it. For the following details of the process I am indebted to the papers of Mr. Charles Butters and Mr. A. Von Gernet, read before the Chemical and Metallurgical Society of South Africa, and published in the South African *Mining Journal.*

**Discovery of the Process.**—Mr. Von Gernet said the electrical precipitation of gold extracted from ores by cyanide had been in use in Europe and Asia as far back as 1888. In 1887, Dr. Siemens found that the gold anodes used in electroplating at his works in Berlin lost weight when standing idle in the cyanide solution, without any electric current passing through the bath. This being backed by the well-known fact that gold was soluble in aqueous solutions of cyanide, first induced him to try the use of that solvent for the extraction of gold from ores.

In the same year, he built a small plant to make experiments on concentrates produced at Siebenburgen. The gold was precipitated both by electrolysis and zinc filings. It was found, however, that the zinc method gave good results only from comparatively strong solutions, while the electrical precipitation was effected with both dilute and strong solutions, and its efficiency was not affected by the presence of caustic soda.

Dr. Siemens therefore decided to use electrolysis only, and early in 1888 he commenced operations on a large

scale. Engineers were sent to different countries, two going to Hungary, one to America, and one (Mr. Von Gernet) to Siberia.

The operations were generally successful, and in May, 1894, a plant, capable of treating 3,000 tons of tailings per month, was erected at the Worcester mine, near Johannesburg. During 1895 the process was adopted by some eight or ten large mining companies, including the Metropolitan, May Consolidated, Crœsus G.M. Co., No. 4 Central Works, and Robinson Slime Works; and already it is a formidable rival of the MacArthur-Forrest zinc-precipitation process.

**Action of the Electric Current on the Cyanide Solution of Gold.**—The electric current decomposes the auro-potassic solution, depositing the gold on the negative pole and liberating the metalloid at the positive pole. In a fixed time a given electric current will deposit a certain quantity of metal, which quantity will vary for different metals in direct proportion to their electro-chemical equivalents. This law holds good only for solutions strong in metal; but with very dilute solutions, as in use in the cyanide process, the current does not find sufficient of the metallic compound present at the electrodes, and consequently decomposition of water also takes place; for this reason, to make the precipitation as efficient as possible, constant diffusion of the solution is required.

The artificial circulation of the solution is most economically and conveniently obtained by allowing a slow but steady flow through the precipitation boxes. It is of the highest importance to give a very large surface to the electrodes, since a more efficient precipitation is obtained by doubling the number of plates than by increasing the current tenfold.

**The Cathode or Negative Electrode.**—To obtain a satisfactory cathode, a metal must be used which will fulfil the following conditions:—

1. The precipitated gold must adhere to it.
2. It must be capable of being rolled out into very thin sheets to save unnecessary expense.
3. It must be easy to recover the gold from it.
4. It must not be more electropositive than the anode, in order to prevent return currents being generated when the depositing current is stopped.

The most suitable metal was found to be lead, which, in the form of lead-foil, meets all the requirements, and is therefore used in the Siemens-Halske process.

**The Anode or Positive Electrode.**—The requirements of the anode are no less important. By the action of the current a metalloid is liberated at the positive electrode, and the latter, when a metal, begins to oxidize. Carbon could be used, but it will not withstand the action of the current, soon crumbling into a powder which decomposes potassium cyanide. Besides, when this finely-divided carbon is in suspension, it cannot be removed from the solution by filtration.

When zinc is used as an anode, it forms a white precipitate of ferro-cyanide of zinc by the reaction of zinc oxide upon ferro-cyanide formed during leaching. In the same way, iron anodes form Prussian blue by the reaction of oxide of iron and ferro-cyanide. In consequence of this reaction, the amount of ferro-cyanide in the cyanide solution does not increase.

The cyanide can be recovered from the Prussian blue formed at the iron anodes, by dissolving it in caustic soda,

then evaporating the solution, and finally smelting with potassium carbonate. Mr. Von Gernet states that this process has been tried on a small scale, about 50lbs. at a time, with the result that a nice clean potassium cyanide was obtained. In the treatment of clean tailings, this regeneration of cyanide is not of great importance; but with concentrates, or trailings which decompose the cyanide solutions with formation of ferro-cyanide, it will effect a considerable saving.

**Electric Current Required for Precipitation.**—Only a very weak current is required to precipitate the gold from cyanide solutions, a density of 0·05 ampère per square foot being sufficient. With cathodes 1½in. apart, 7 volt is sufficient to produce this strength of current.

The advantages gained by using such a weak current are:—

1. The gold is deposited hard on the lead-foil.
2. The iron anodes are preserved for a long time, as their waste is in proportion to their current strength. In a plant treating 3,000 tons per month, 1,080lbs. of iron are destroyed in that period.
3. Little power is required. 746 Watts equal one-horse power. A 3,000-ton plant requires 2,400 Watts, equal, theoretically, to 3½-horse power, and actually requiring about 5-horse power.

**The Advantages of Electric Precipitation.**—The principal advantages claimed for this process are as follows:—

1. That the precipitation operates independently of the amount of cyanide or caustic soda present in the solution. Therefore, in the treatment of tailings,

very dilute solutions can be used, the only limit being a sufficient quantity of cyanide to dissolve the gold satisfactorily.

A solution containing 0·03 per cent. of cyanide will dissolve gold as effectively as a solution containing 0·3 per cent., provided a longer time is allowed for treatment. In the first case, the decomposition of cyanide is much less than in the second, resulting in a corresponding economy.

2. However acid the solution may be when entering the extractor, the precipitation takes place equally as well when the solution is neutral or alkaline.

3. No complications arise from the formation of lime, alumina, or hydrate of iron, which sometimes cause trouble in the zinc process of precipitation.

4. With ores or tailings containing copper, the extraction of the gold will be the same, but the decomposition of cyanide less than when using stronger solutions.

5. The successful treatment of slimes.

**The Actual Working of the Process.**—The first practical demonstration of this process on a commercial scale took place at the cyanide works of the Worcester Gold Mining Company, near Johannesberg, under the supervision of Mr. A. Von Gernet.

The plant consists of five leaching vats placed on a row of stone piers, with a single tunnel beneath. Each vat is 20ft. in diameter, with 10ft. staves, and has a capacity of 100 tons of tailings.

Between the vats and the electric extractors there are placed two tanks, 16ft. in diameter, with 6ft. staves, forming

two intermediate reservoirs, which enable the flow through the precipitation boxes or extractors to be kept constant and steady, a matter of great importance.

A better method to secure a even flow is to pump all the solution into a small raised tank, provided with an overflow into the intermediate tank, and a delivery pipe to the precipitation boxes. The small tank is always kept full to overflowing, so that it delivers under a constant hydraulic head.

Beyond the precipitation boxes there are two sumps, 20ft. in diameter and 6ft. deep, from which the cyanide solutions are returned to the leaching vats.

Two collecting vats, 20ft. in diameter and 8ft. deep, receive the tailings from the 25-stamp battery.

**The Electric Precipitation Boxes.**—There are four precipitation boxes, constructed of wood, each 18ft. long, 7ft. wide, and 4ft. deep. Each box contains 89 iron-plate anodes, 7ft. by 3ft. by $\frac{1}{8}$in., cased in canvas to retain the small quantity of Prussian blue produced; and 88 cathodes of lead-foil stretched on iron wires fixed on a wooden frame. Each frame contains three strips, 3ft. by 2ft., so that, counting the double surface of each lead sheet, there are altogether about 3,000 square feet of cathode surface, the current density being 0.05 ampère per square foot. Copper wires are fixed along the top of the sides of the boxes, and convey the current from the dynamo to the electrodes.

The boxes are made of 3in. material throughout, with stiffening pieces across the sides and bottom. The divisions are of wood, or are formed by raising some of the iron plates about an inch above the level of the solution, while

others rest right down on the bottom, the joints being made watertight by means of wooden fillets caulked with hemp packing. By this means a series of compartments is obtained, similar to those in a zinc precipitation box, the difference being that the solution passes alternately up and down through successive compartments. The rate of flow is about one foot per minute.

**The Clean up.**—The boxes are kept locked, being opened once a month for the "clean up," which is conducted as follows:—The frames are taken out singly, and the lead-foil is removed and replaced by fresh lead-foil, the whole operation taking but a few minutes for each frame. The lead, which contains from 2 to 12 per cent. of gold, is then smelted into bars and cupelled.

The gold is deposited on the lead sheets as a thin bright yellow film which adhers firmly to the lead. The consumption of lead at the Worcester Works is 750lbs. per month, equal to 1½d. per ton of tailings.

The working expenses for treating 3,000 tons per month were as follows:—

| | D. | |
|---|---|---|
| Filling and discharging leaching vats .. | 10·00 | per ton |
| Cyanide, ¼-lb. .. .. .. | 6·00 | ,, |
| Lime .. . .. .. | 1·20 | ,, |
| Caustic Soda .. .. .. .. | 0·50 | ,, |
| Lead .. .. .. .. | 1·10 | ,, |
| Iron .. .. .. .. .. | 2·20 | ,, |
| White Labour .. .. .. | 5·20 | ,, |
| Native Labour and Food .. .. | 1·90 | ,, |
| Coal .. .. .. .. | 4·60 | ,, |
| Stores and General Charges .. .. | 3·30 | ,, |
| Total .. .. .. | 36·00 | per ton of 2,000lbs. |

The cost of treatment per ton of 2,240lbs. would be 3s. 4·32d.

The tailings assayed from 6dwts. to 8dwts. of gold, and the residues, after treatment, from 1dwt. to 2dwts. per ton. The average actual extraction was about 74 per cent.

The solutions, after leaving the precipitation boxes, still contained gold, the strong solution showing by assay 4dwts. 8grs., and the weak solution 10grs. per ton of solution. On the average, the strong solutions carried from 4dwts. to 5dwts., and the weak from 0dwt. to 1dwt. of gold per ton of solution.

From November, 1894, to May, 1895, the Metropolitan Company treated 26,900 tons of tailings for 4,845oz. of gold, at a cost of 2s. 8d. per ton. At the May Consolidated the working expenses amounted to about 2s. 4d. per ton, excluding the royalty for the use of the process, which amounts to 3 per cent. The extraction amounted to over 80 per cent. of the original assay value.

**Details of the Treatment.**—The time occupied in leaching and washing, together with the quantity of the solutions, are given in the following tabulated statement:—

| | HOURS. |
|---|---|
| Alkaline wash, 10 tons .. .. .. .. | 3 |
| Strong cyanide solution, 70 tons, 0·05 to 0·08 % KCy, applied in 14 separate portions of 5 tons each .. | 65 |
| Weak cyanide solution, 21 tons, 0·01 % KCy, applied in 3 portions of 7 tons each .. .. .. | 18 |
| Water washes, total 11 tons, pumping dry and discharging .. .. .. .. | 22 |
| Total .. .. .. .. | 108 |

The working of this process gives rise to the production of a number of valuable commercial bye-products, including copper, lead, litharge, and paint.

## CHAPTER XI.

### OTHER CYANIDE PROCESSES.

**The Hannay Electro-Cyanide Method.**—In this process, the pulverized ore is agitated in an iron pan with a dilute solution of cyanide. While the agitation is going on an electrical current is passed through the charge in the pan. To effect this, there is a bath of mercury at the bottom of the pan. This forms the cathode or negative electrode.

Several inches above the level of the mercury-bath there is fixed an annular ring, composed of a mixture of powdered graphite and resin, compressed into a solid mass. This forms the anode or positive electrode.

In this process it is claimed that the coarser particles of gold are amalgamated with the mercury, while the finer and more refractory particles are quickly dissolved by the cyanide and instantly deposited in the mercury by the action of the electric current, supplied by a dynamo. By this means the whole of the gold is obtained as an amalgam.

The treatment is said to take from two to four hours, according to the nature of the ore. A number of working tests were made in London, with very satisfactory results. From a pyritic ore from Borneo, 89·2 per cent. of the gold, and 88·5 per cent. of the silver were extracted. An ore from Marototo, New Zealand, said to contain much

tellurium and selenium, yielded 95·3 per cent. of the gold and 98·3 per cent. of the silver. A test of 10 tons was made at one time.

The inventor claims that the electro-cyanide process only attacks metals and silver salts, so that copper in an ore, when existing as pyrites, is not attacked, but is stripped of its gold just the same as iron pyrites.

This process seems to be an improved amalgamation process, and should be capable of useful application for the treatment of pyritic concentrates and suitable ores.

### THE PARK-WHITAKER CYANIDE PROCESS.

This process is intended for the treatment of cupriferous ores and concentrates which cannot be treated successfully by the ordinary cyanide processes, on account of the solubility of copper ores in cyanide solutions.

In this process the ore is subjected to a chloridizing roasting, after which the soluble copper chlorides are removed by leaching with water. An alkaline wash is then applied, and the gold and silver extracted with a dilute solution of cyanide.

During the roasting the silver sulphides present are chloridized to the chloride, which is readily dissolved by cyanide. The dissolved copper is recovered by passing the solutions through iron turnings or scrap-iron.

Experiments on a working scale were made by the author on a parcel of ore from the Monowai mine, N.Z., with most successful results, and preparations are now being made for more extensive trials.

## CHAPTER XII.

### ANTIDOTES FOR CYANIDE POISONING.

All cyanides are deadly poisons; but the aqueous solutions used in practice are so dilute that there is little or no danger from the prussic acid evolved from them if the buildings are properly ventilated.

Acids react on cyanides, liberating prussic acid gas, which causes almost instant death when inhaled in a pure state. When diluted with air, it causes faintness, dizziness, and a depressing frontal headache.

Even very dilute solutions of cyanide are poisonous when taken internally; and, when they come in contact with the skin, produce, in some persons, an eruption of painful red boils. In cases where the hands and arms must be brought into contact with the solutions, rubber gloves. reaching over the elbows, should be provided for the workmen. Kaffir workmen are said to suffer no inconvenience whatever from the contact of their skin with cyanide solutions.

Considering the extensive use of cyanide, the number of fatal accidents is remarkably small. Up to the present time only one fatal case has been recorded in New Zealand.

In case of accident from cyanide poisoning, the following remedies are recommended:—Put the patient into a hot bath, and apply cold water to his back and neck. In cases of internal poisoning, vomiting should be induced by emetics, or by physical means.

Freshly precipitated carbonate of iron, obtained by mixing equal quantities of sodium carbonate and ferrous sulphate, is recommended for internal use.

If the poisoning is the result of inhaling prussic acid gas, it is advisable to make the patient inhale a small quantity of chlorine gas, ammonia, or ether. The chlorine gas can be quickly made, and applied by sprinkling a little bleaching powder on a piece of flannel moistened with acetic acid, and then holding the flannel to the nostrils of the patient.

It was lately reported in the press that Johann Antal, a Hungarian toxicologist, had found that a solution of cobalt nitrate was a perfect antidote for prussic acid poisoning.

# MISCELLANEOUS.

## QUESTIONS USED IN EXAMINATION OF BATTERY SUPERINTENDENTS FOR CERTIFICATES.*

FIRST DAY.—TIME: 9 A.M. TO 1 P.M.

### Subject A: The Different Modes of Reducing and Pulverizing Ores.

1. Show by sketch how you would construct the foundations for a stamp-mill with twenty heads of stamps, and give description of same.

2. What weight of stamps do you consider best suited for dry-crushing? Give their drop, and number of blows per minute; and show by calculation the theoretical horse-power required to drive a battery of twenty heads of such stamps.

3. In constructing a new stamp crushing-battery for dry-crushing, and the ore to be treated by the cyanide process, what mesh and description of screens would you use, and at what height in relation to the bottom of the screens would you place the dies in the mortar?

4. What depth of ore would you keep in the mortar to produce the greatest crushing efficiency with economy? and give your reasons why.

5. Describe the action of Blake-Marsden and Gate's rock-breakers, the speed they should be worked at to give the most efficiency with economy; also the dimensions that the ore should be reduced to by rock-breakers before going into a stamp-mortar.

6. Describe the action of Cornish and Krom rolls; also a Huntingdon mill: and show by sketch how you would screen the ore if rolls were used, and how you would dispose of the coarse particles that did not go through the screen.

* Questions used at N.Z. Government May Examinations, 1896.

7. Describe the Mundy and Kaiser buddles; give their dimensions, and speed they should be worked at to produce the best effect; also the angle at which the bottom should be constructed in relation to the horizon.

8. If you were using a Challenge ore-feeder, describe how you would fix it to feed the stamps automatically.

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### Subject B: Amalgamating-machines.

1. Describe the action and speed of the following grinding and amalgamating-machines—namely, Combination, Watson-Denny, and McKay pans; the quantity of tailings each pan would manipulate efficiently in every twenty-four hours; also the theoretical horse-power required to drive each pan.

2. Give a sketch of a separator or settler, showing its dimensions; also give the speed it should be worked at, and the time required for the manipulation of each charge.

3. What is meant by agitators, what is their use, and how are they constructed and worked?

4. In using a plant of twenty berdans for grinding and amalgamating, what quantity of tailings would this plant be capable of treating in twenty-four hours? Give the speed at which you would propose to work the berdans, and the quantity of quicksilver you would use in each berdan; also the theoretical horse-power required to work the plant.

5. Give a detailed description of the Washoe process from the time the ore is put into the pans until the bullion is retorted.

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FIRST DAY.—TIME: 2 P.M. TO 5 P.M.

### Subject C: The Use of Quicksilver and the Methods of Using it in connection with the Extraction of Gold and Silver from Ores.

1. What effect has ore containing galena, zinc-blende, and antimony on quicksilver?

2. How would you coat copper plates with quicksilver? Also describe how you would remove all the gold and silver from plates that had been used for amalgamating purposes.

3. What quantity of quicksilver would you use in a Watson-Denny pan?

4. How would you detect impurities in quicksilver, and remove them?

5. Describe the classes of ore most suitable for amalgamation with quicksilver, and give your reason why.

6. What chemicals, if any, would you use in pan-amalgamation? and, if used, give your reasons for same, and also their reactions.

7. Describe the method you would adopt for recovering the gold and silver from ores containing the sulphides of copper, lead and antimony.

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Subject E: Chlorination Process of Recovering Gold from Ores.

1. Describe the class of auriferous ores best adapted for treatment by chlorination, and give your reasons why.

2. Why do pyritous ores require roasting before chlorination? and describe the process of roasting such ores in a reverberatory furnace.

3. How is chlorine gas manufactured? and how is it applied in the extraction of gold from ores?

4. What time is required in subjecting the ore to the action of chlorine gas by the Plattner's process, and also by the Newbery-Vautin process?

5. Give a sketch of a modern chlorination plant, and mark each portion with distinctive letters; also give the time occupied in each operation with the ore.

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SECOND DAY.—TIME: 9 A.M. TO 1 P.M.

## Subject D: Cyanide Process of Recovering Gold and Silver from Ores.

1. Describe how you would make silver-nitrate standard solutions to test solutions of KCy; also show how you would test such solutions.

2. Describe how you would test the strength of crude KCy.

3. If a vat is 22ft. 4in. in diameter, and contains a depth of 19in. of ore, how many cubic feet of ore is there in the vat, and what weight of solution will occupy 10in. in depth of such vat?

4. Show by sketch a complete plant for treating pulverized ore by the cyanide process, and how you would arrange it; also mark each portion of the plant by distinctive letters, and state what the letters refer to, giving the dimensions of the different parts.

5. How many pounds of a strong solution containing 18 per cent. of KCy should be used to make up 11 tons of a 0·35 per cent. solution, using a sump solution containing 0·11 per cent. of KCy.

6. How many tons of 0·5 per cent. solution can be obtained from 13 tons of 0·9 per cent. solution, using a 0·17 per cent. solution for dilution?

7. In leaching auriferous or argentiferous ores with potassium-cyanide solutions, if such ores contained sulphides of copper, antimony, and zinc, what effect (if any) would each of these metals have on the extraction of the gold and silver from the cyanide solutions?

8. Describe the best method of preparation of zinc used for the precipitation of the gold and silver from cyanide solutions, and give the reasons why.

9. Describe how you would assay sump KCy solutions to ascertain the quantity of gold and silver they contain.

10. If you had to treat ore by the cyanide process containing 80 per cent. of fine gold and 20 per cent. of coarse gold, what means would you adopt to recover the latter?

11. If you have an 18 per cent. strong solution of KCy in the dissolving-vat, and you required 15 tons of a 4 per cent. working solution, how many pounds of the strong solution would you use to make up the required quantity, first utilizing 2·5 tons of a 1·12 per cent. solution in the reservoir?

12. What was the average consumption of cyanide and zinc per ton of ore in any cyanide works you have been connected with?

13. What are the antidotes of prussic acid? If a workman showed signs of cyanide poisoning what remedies would you apply?

14. What quantity of KCy is required for the solution of 1oz. of gold and 1oz. of silver per ton of ore? and how does this quantity compare with the theoretical quantity?

15. In treating poor tailings is there any limit to the weakness of the solution used when zinc is the precipitant, and why? Also, what substitute for zinc can be used as a precipitant when the solutions are very weak?

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SECOND DAY.—TIME: 2 P.M. TO 5 P.M.

### Subject F: The Sampling and Testing of Ores.

1. Give the principal reactions of both the oxides of iron when in solution.

2. Name the physical characteristics of zinc-blende, and the methods by which you would chemically determine it.

3. State the methods for detecting antimony in siliceous rocks.

4. How would you detect minute quantities of copper in lode matters?

5. What would be your easiest and most rapid method for estimating the proportion of copper in a mixed sulphide?

6. Name the tests for carbonic acid in both air and rocks.

7. How would you test pyritous quartz for silver and gold?

8. What process would you use for detecting silver in its ores?

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### Subject G: A Knowledge of Arithmetic and the Method of Keeping Accounts

1. Divide ·0654 by 3·145, and extract the cube root of the quotient.

2. If seven men performed a certain piece of work in 3 days $2\frac{1}{2}$ hours, how long would it take a man and a boy to do the same work, the boy being able to do five-eighths of a man's work?

3. If 1oz. of gold of 24 carats is worth £4 4s. 11d., what would be the value of 15oz. 11dwt. of gold of 15·875 carats fine

4. A circular vat is 19ft. 6in. in diameter and 3ft. 3in. deep; give its cubic contents in feet.

5. Give the superficial board measurement in feet of four 17ft. engths, 10in. x 1in., three 19ft. lengths, 8in. x 3½in., and seven 20ft lengths, 17in. x 2½in.

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